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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

MINING PRACTICE AT MORENCI BRANCH, PHELPS DODGE CORPORATION, MORENCI, ARIZ.

BY McHENRY MOSIER AND GERALD SHERMAN

CONSULTING ENGINEERS

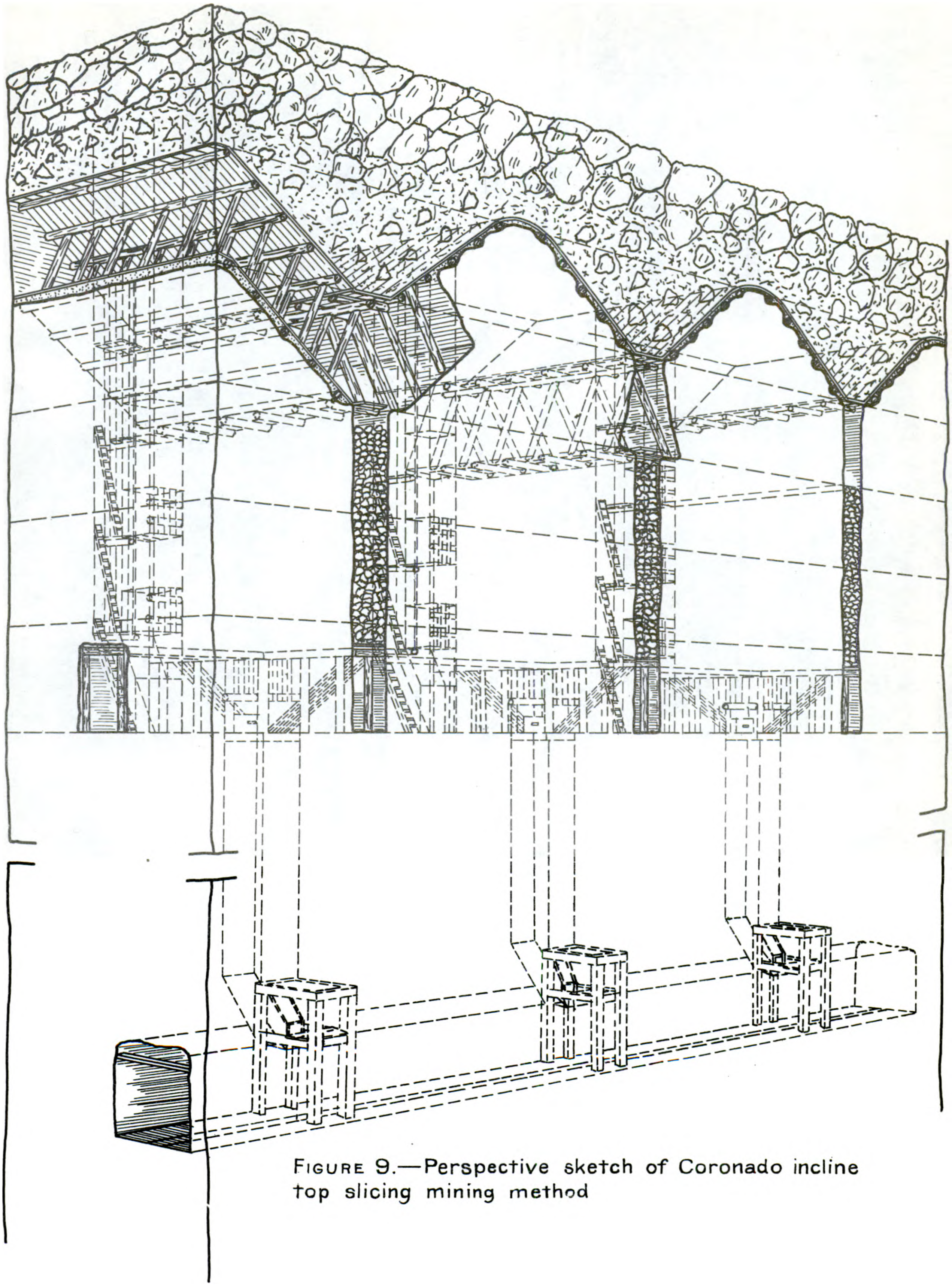

FIGURE 9.—Perspective sketch of Coronado incline top slicing mining method

Washington, D. C.
March, 1929.

This paper is the first of a series of publications dealing with the caving method of mining and is to be followed by similar papers on other mines in Arizona and in other districts throughout the United States, where this system is used.

At the same time that these reports are being prepared for publication, and there will be a dozen or more, material is being collected for a general bulletin presenting the caving system of mining, its application and limitations under various conditions, with discussions of the problems encountered and their solutions.

These papers are all being written by officials and engineers of the mining companies in accordance with an outline prepared by the Bureau of Mines for the purpose of obtaining uniform and comparable data.

Scott Turner

SCOTT TURNER,
Director.

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING PRACTICE AT MORENCI BRANCH, PHELPS DODGE CORPORATION, MORENCI, ARIZ.[1]

By McHenry Mosier and Gerald Sherman[2]

INTRODUCTION

This paper describing mining practice at Morenci, Ariz., is one of a series of articles on mining methods and practice in the various mining districts in the United States.

At present (1929) practically all the mineral production in the Morenci district is from the Humboldt mine of the Morenci Branch, Phelps Dodge Corporation. The mining method used is adapted to the type of ore body being exploited; it has been developed by adopting methods used elsewhere and making necessary changes to fit local conditions. Improvements in current practices are still being made.

ACKNOWLEDGMENTS

The authors acknowledge the assistance of L. F. Hersey, chief engineer, mining department, and C. P. Burford, engineer in charge of stope control, both of the Morenci Branch; and of E. D. Gardner, supervising engineer, and C. H. Johnson, assistant mining engineer, United States Bureau of Mines at Tucson.

HISTORY OF THE DISTRICT

The Clifton-Morenci district is in southeastern Arizona on the southern slope of the Gila Range of mountains, near the junction of the Gila and San Francisco Rivers. Its elevation is 4,800 feet. Clifton is about 69 miles by rail northwest of Lordsburg, N. Mex., on the Southern Pacific Railroad. Morenci is served by a narrow-gauge freight railroad from Clifton (fig. 1).

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[2] Consulting engineers, U. S. Bureau of Mines.

Gold was discovered in 1872 by a party of prospectors from Silver City, said to have been connected with the U. S. Army. Copper ore indications were noted on Copper Mountain, and several claims were staked, for copper ore but because of the lack of rail transportation in the early days, the first real interest was in gold. The first copper smelting began in 1873 when a cupola furnace having a capacity of 1 ton per day was built, but only the very richest ores could be treated.

The railroad was built through Lordsburg in 1881, and a branch line was extended to Clifton in 1884. The building of the railroads stimulated mining in the district by decreasing operating costs. In 1882 both the Detroit Copper Co. and the Arizona Copper Co., Ltd., were organized. In 1900 the Shannon Consolida Copper Co. began operations.

The first smelters in the district treated copper oxide ores mined from sedimentaries near the main porphyry contact. In 1893 chalcocite ore bodies wer opened in the Copper Mountain porphyry. As the rich oxide ores became depleted, attention was directed to the lower-grade ores in the porphyry. The first conce trator was built in 1886.

In 1922 the Arizona Copper Co., Ltd., which had previously absorbed the Shannon Copper Co., merged with the Phelps Dodge Corporation, whose Morenci Bran was originally the Detroit Copper Co. Important economies in operations were ma possible by this consolidation for the following reasons: (1) On account of con tiguous ore bodies, which are now being mined as single units by cheaper methods at a more intense rate; (2) by milling all ores at one concentrator, remodelled treat efficiently the present ores; (3) by smelting ores and concentrates at one modern plant; and (4) by reducing overhead and certain operating costs through t elimination of duplication, as one organization now serves in the place of two.

In the production history of the district there have been three periods:

(1) From the discovery of the mines until the construction of the railwa During this time only the rich oxidized ores from the sedimentary rocks and from surface veins in the porphyry could be mined and smelted on the ground.

(2) From the construction of the railway in 1884 until about 1915. This covers the period of the concentration of comparatively rich sulphide ores while the production of direct smelting ores still continued in important quantities.

(3) Since about 1915 the smelting ore has shrunk to insignificant propor tions, and the production has been made from secondarily enriched, but leaner, sulphide ores from unoxidized areas in the porphyry.

PRODUCTION

The annual production from the district attained its peak in 1916, with 70,000,000 pounds of copper. At present the output is almost 60,000,000 pounds year; it is produced by concentrating 1,500,000 tons and by smelting a small tonnage of furnace ores and converter flux. To the end of 1927 the district had produced a total of approximately 1,800,000,000 pounds of copper.

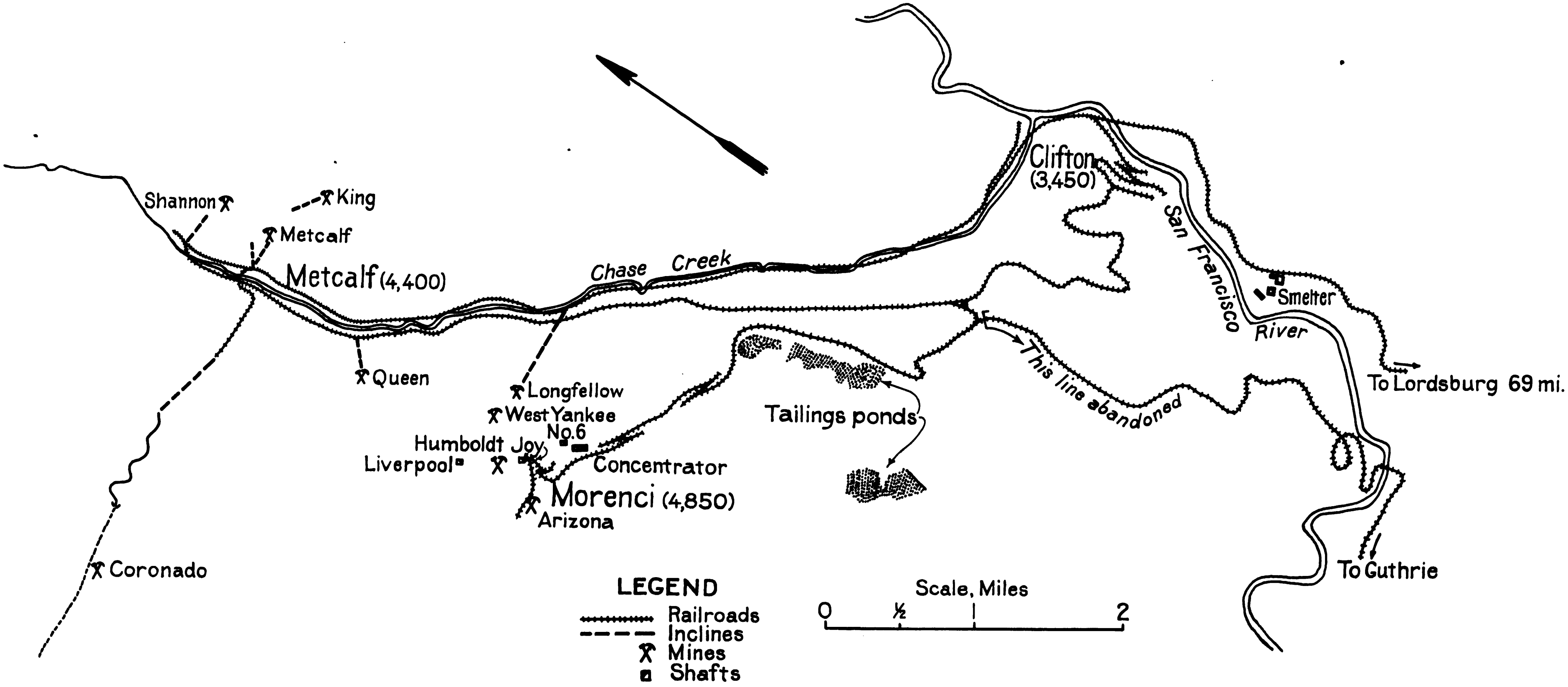


FIGURE 1.—Clifton – Morenci District

GENERAL GEOLOGY OF MORENCI DISTRICT

In the Morenci district, as distinguished from the Metcalf and Coronado deposits, copper mineralization was introduced with or after the intrusion of a large mass of porphyry below a series of sediments.

Rich, irregular ores, often oxidized, were found in the limestone and shales adjoining the porphyry contact. Surface veins of good ore also outcropped on the exposed surface of the porphyry. These ores supported the mines during the early days. The large deposit of concentrating ore at the Humboldt mine, which now is being worked, lies deeper in the porphyry and is localized by major systems of faults and fractures.

The deposit is comparatively shallow and lies wholly above the level of the deep channel of Chase Creek which passes a short distance to the northeast.

The Humboldt Mine

The Humboldt mine, which was the principal source of copper of the Arizona Copper Co. and is now of the Morenci Branch of the Phelps Dodge Corporation, is in the center of the town of Morenci at the foot of Copper Mountain. The mine has been worked almost continuously for nearly 40 years.

The country rock at Morenci is a quartz monzonite porphyry and a series of sedimentary rocks consisting of limestone, shale, and quartzite generally metamorphosed. The porphyry is intrusive and occurs as a large mass and also as dykes in the sediments in the contact zone.

The ore bodies of the Humboldt mine are in the main porphyry area a few hundred feet northwest of the contact with the altered sediments. The porphyry contains many small detached bodies of metamorphic limestone and shale. The porphyry has a close structural and genetic connection with the ore deposits; a porosity and looseness of grain have made it easily permeable to ore-bearing solutions. Fresh rock occurs rarely, and a large part is completely altered to sericite, quartz, and pyrite, chalcocite being usually associated with the latter mineral.

Chalcocite is the only important ore mineral in the mine. In the upper levels of the chalcocite zone, green films of malachite and brochantite are commonly found. In this mine the chalcocite gives clear evidence of being secondary and occurs deposited on pyrite, beginning as slight black films and gradually replacing the mineral entirely.

Fissures cut through the metamorphic rock as well as the porphyry, but in the former they are rarely productive, the softened porphyry being evidently the most favorable ground for ore. The rich seams consist of pyrite, containing more or less chalcocite but very little quartz. Frequently they are bounded by well-defined planes with evidence of movement and also may be traversed by slips parallel to the walls. The wall rock immediately adjoining the rich seams is, in some cases, converted to almost pure kaolin. This mineral, however, does not exist as a constitutent of the rock. On both sides of these seams extends a mass of sericitized porphyry, of varying widths, containing little seams and grains of chalcocite.

The ore body now being mined is in zones of fracturing and has no definit walls. The limit of the ore is determined by assay. Irregular fracturing occur in three directions, but the fracture planes have been more or less recemented with quartz and pyrite. The planes of weakness run from 2 to about 18 inches apart.

Permanent drifts and raises in the ore body have to be timbered. Temporary workings generally stand without timber.

The present ore reserves lie principally in one mass in the porphyry, striking irregularly N. 25° E. along two intersecting fault zones through a maximum length of 2,000 feet and a maximum width of 600 feet. The vertical range is about 1,000 feet. The ore body, throughout the northern portion where it follow the Copper Mountain fault, dips 65 to 75° to the eastward, while at the middle a to the south, along the Fairplay fault, it is nearly vertical. The ore body pitches northward along the Copper Mountain fault.

Much of the porphyry wall rock carries at least 0.4 per cent copper. It is generally stronger than the ore physically, because it is less altered, and carries less kaolin and sericite.

Above the 5th level the southeastern edge of the ore body lies against t main porphyry-sedimentary contact, while below that level and to the northward t ore lies at an increasing distance from the contact. The contact is highly silicified and consequently the ore along it is hard and breaks coarsely. Conversely, ores with increased alumina break more finely.

The capping is occasionally soft hematite, but this material usually con tains enough silica to prevent packing, as is indicated by the gob in old square set stopes and by the waste, following down after former top-slice stopes.

The present lift of the ore body being worked is situated for the most part either (1) under top-slice stopes with overlying timber mats, or (2) below sub-level caving stopes immediately beneath those top slices. The timber mats a of no assistance in preventing dilution and, furthermore, cause delays at the grizzlies and even in the chutes below. The timber also interferes with flotati

Although the ore in the Humboldt mine is somewhat harder than at other porphyry mines in the Southwest its caving is made possible by the major faults with their sympathetic sheeting, by the universal jointing throughout the ore bo by the minute fracturing of the ore in all directions, and by the alteration of the ore.

The grade of the ore reserves is about 1.90 per cent total copper with per cent acid-soluble copper. Near the bottom of enrichment the ore is slightl higher grade. In depth it carries somewhat more alumina, and the acid-soluble copper content decreases.

The average composition of the ore going to the mill is:

	Per cent
Silica	65
Alumina	17
Potash	5

The ore contains 3.5 per cent moisture as it leaves the mine. Due to sericite, the alumina content of the ore is unusually high, and when finely-divided ore is wet it tends to stick in the chutes and to hang up in raises.

METHODS OF PROSPECTING AND EXPLORATION

In the exploration and development of the large and comparatively regular low-grade Humboldt ore body the first work was unsystematic and was done in the search for high-grade ore. When the present low-grade ore became of importance, information gained in the former work was used and supplemented by systematic work to prove the assay grade, size, and outline of the ore body.

The thorough exploration of ore deposits to be caved is necessary because operations must be planned in advance for the whole operation. A large part of the cost of mining is incurred before production begins, and plans can not be changed to suit unexpected conditions.

The Humboldt ore was explored entirely by underground drifts and raises. Churn drills have been used elsewhere in the district, and a separate ore body is now being explored by diamond drills.

METHODS OF SAMPLING AND ESTIMATING TONNAGE AND GRADE

Underground openings, such as drifts and raises, are sampled by cutting channels in the ore with hand hammer and moil. Care is taken to cut the channels perpendicularly to the vein structure. As the porphyry is cut by numerous tiny veins, the channels have no regular directions relative to the workings. Some are parallel to the floor about waist high; others are cut diagonally across the back. Each sample represents 10 feet of drift or raise.

In diamond drilling, 50 per cent of the core is recovered which does not truly represent the ground drilled. This has made it necessary to save all the core and a known fraction of the sludge from every sampling interval. These are combined according to weights and grades to obtain the average grade of each sample. The sludge is split, to deliver one-eighth of the water and rock from the hole, with a splitter especially constructed for the work. The resultant sludge is allowed to settle in barrels until the water is clear. The water is then decanted off and the sludge evaporated to dryness in an electric drying oven. The dried sample is ground, mixed, and assayed. Tests are made periodically of the water decanted from the sludge.

To obtain the final assay for any sampling interval the following procedu is used: First, the core is weighed and put through the crusher and rolls in the sample mill. The resultant pulp is carefully mixed and split to a 3-ounce sample which is ground to -200 mesh and sent to the assay office for electrolytic assayi for total and acid-soluble copper. The reject is placed in an air-tight can for possible future reference. Second, the sludge is dried in electric dryers, weigh and carefully mixed and split to give another 3-ounce sample for pulverizing and assaying similarly to the core sample. The remaining pulp is also filed in an ai tight can. Third, a known percentage of the water decanted from the sludge is evaporated to dryness, weighed, and assayed to check the possible loss of sludge held in suspension and carried off by the water rejected. Fourth, the average copper content of the sample is determined by combining the assays of the core an sludge in proportion to their weights. If the weight of the sample does not correspond with the calculated weight of the section of the hole drilled the results may not be given full weight. Adjustments are made in using these results as well as the results of individual samples that are unusually high in copper.

For the estimation of ore reserves a full knowledge of the ore deposit mu be obtained. Caving stopes have reasonably regular outlines, and selective minin is therefore not practicable by this method.

Some material of a grade that will not pay to reduce must be mined, and some good ore on the boundaries must be left because its inclusion would bring i too much waste. The side boundaries, which are vertical or nearly vertical, are drawn as compromise planes to inclose as much ore as possible without too much waste.

Except for preliminary estimates, the volume of material within the stop outlines constitutes the ore reserves which are bounded by (1) the undercutting level, (2) the shrinkage side outlines, and (3) the leached gossan or a stope ab as the case may be. Within these boundaries the grade of ore in place is calculated by combining assays in a rational manner.

DEVELOPMENT WORK

The mine is worked through three shafts and has two adits. Figure 2 is vertical diagrammatic sketch of the mine. Ore only is hoisted through No. 6 sha Supplies and men are handled through the Joy shaft, and the Liverpool shaft is u mainly for handling waste and supplies and men doing development work on the low levels. The main shafts are located at such distances from the ore body that th will not be affected by ground movement due to the caving system of mining in us

The present development system is the outgrowth of former operations in more scattered locations.

The porphyry is fairly uniform in the mineralized section. Due to the planes of weakness the rock is broken readily by explosives.

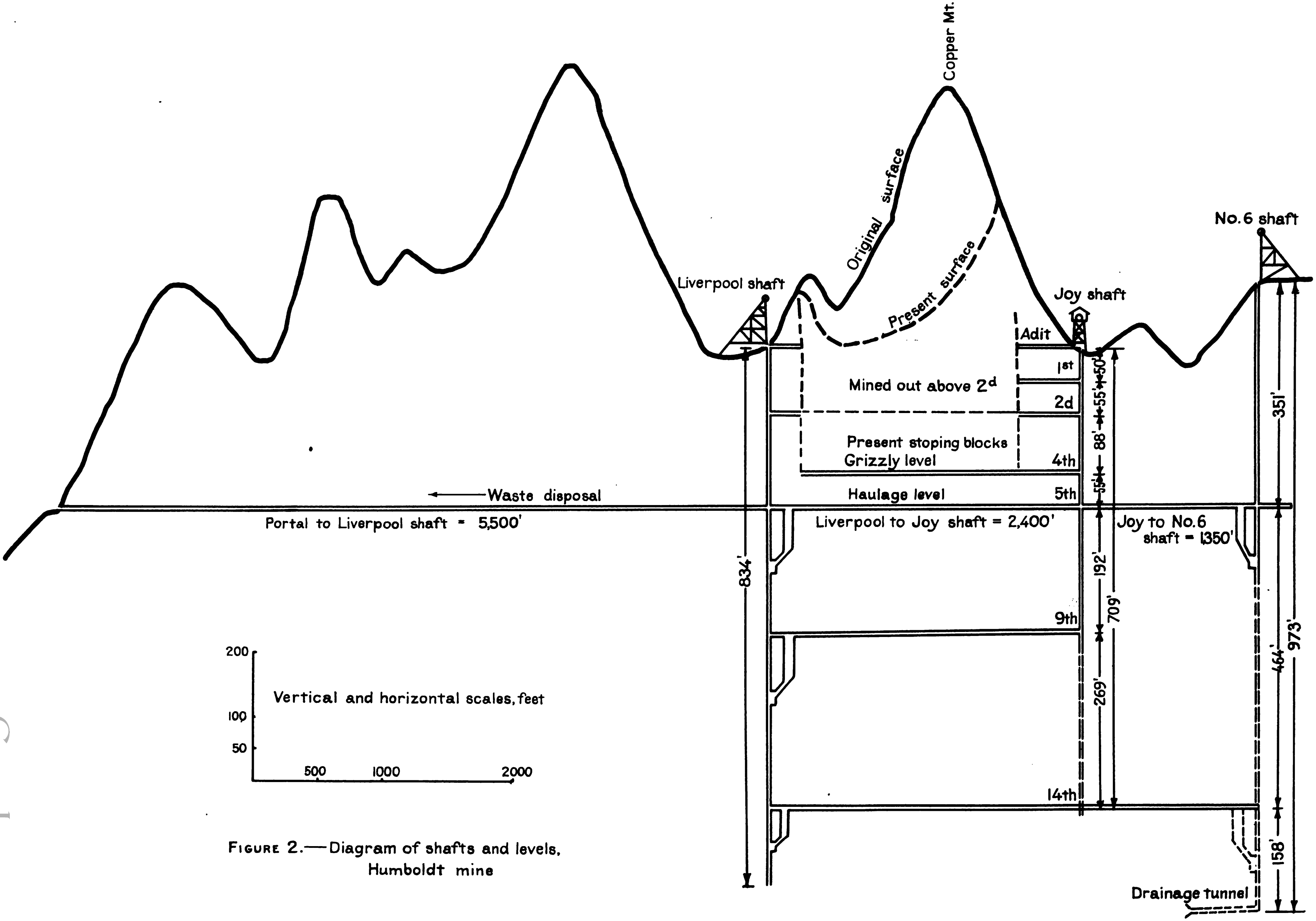


FIGURE 2.—Diagram of shafts and levels, Humboldt mine

The stope development work and the haulage drifts under the stopes are in the porphyry and have been generally in ore. Development work and long haulage drifts are also driven in limestone, quartzite, and granite. The granite through which a drainage tunnel is being driven contains few planes of weakness and is very difficult to break. The other rocks are harder to break than the mineralized porphyry.

Development work is planned far enough ahead so that one operation does not interfere with another. Generally a miner is given two or three faces in which to work. With three faces available, miners frequently drill $1\frac{1}{2}$ to $2\frac{1}{2}$ rounds per shift. In drifting it is desired only to blast as deep a round as can be shoveled out in one or two complete shifts. This leaves a clean set-up for drilling the next round.

Standard rounds for all development work are of the same general type; the modifications are in the number of holes used. A standard is available for any type of ground that may be encountered. The V-cut round is used, and care is taken to make the cut holes meet to insure simultaneous detonation. This is especially necessary when gelatin dynamite is used, because it is relatively insensitive to shock. Cut holes are drilled in the center to take advantage of the most freely-breaking area in the face.

The amount, distribution, and grade of explosive used are also standardized. The standards were obtained after making many experiments in different classes of ground, and it is considered that by the consistent use of the standards set greater economy can be obtained. Occasionally a heading could be broken by a round with fewer holes or with less explosive; however, a few failures are more than offset by the advantages gained. The miners readily learn to drill standard rounds. The bosses advise the men in the selection of the round and specify the quantity and kind of powder to be used.

Practically all development work is done on contract. A uniform price is set for each class of work.

Drilling

Drift rounds in large headings or in hard rock are usually drilled with drifting machines mounted on vertical columns. It is considered that they can be drilled more nearly to the standard by using columns than with crossbars. Small drift headings and some of the larger headings in the easily broken rock are drilled with unmounted jackhammers. In most cases where an unmounted drill is used its weight is supported on a length of drill steel. A small hook is used for attaching the drill to the steel; one end of the hook fits around the end of the steel above the lug and the other end under the frame of the drill. As different lengths of steel are always available at the face, the holes can be started at any desired height.

In average Morenci ground, which drills easily, the jackhammer type drill gives excellent service. In the quartzite and granite occasionally encountered i the lower levels, drifting machines have proved more satisfactory; they drill faster, maintenance costs per foot of advance are less, and fewer pieces of steel per hole drilled are used.

Drills of the jackhammer type are used in sinking shafts and winzes. Raises are run with stopers. Hand-rotating machines are generally employed but self-rotators are used in the hardest ground.

The drill-repair shop is under the direction of the development departmen and spare drills in good condition are always available at tool houses located at convenient points in the mine.

Steel.- Only hollow drill steel is used. Seven-eighths-inch hexagonal steel is used for jackhammers, and the same size of quarter octagon for the stope One-inch round steel is used for drifters in rock other than porphyry. The gauge of starters is 1-15/16 inches; the reduction in diameter for each succeeding bit is 1/16 inch. The difference in length between each change is about 15 inches. Bits are sharpened carefully, so that variations in the diameters of different bi are less than 1/64 inch.

Drill Bits.- A double taper bit having angles of 8 and 14°, with flattene wings, has been developed at Morenci (fig. 3). The porphyry is easily drilled, a the usual type of drill bit will lose its gauge before the cutting edge is dulled It is considered at Morenci that the flat-wing type of bit drills faster with les loss of gauge than the standard form. In soft ground the side-hole bit is used.

Air Pressure.- The air pressure at the compressors is 90 pounds per squar inch and at the drills is 80 to 85 pounds.

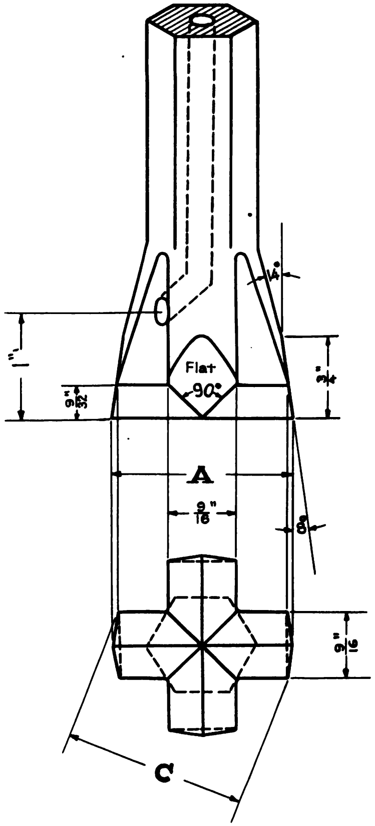
Blasting

Loading Holes.- A standard method of loading holes is followed for all rounds. Primers with fuse and detonators are made up by inserting the detonator in the side of the primer cartridge and securing it in place by tying or taping fuse to the cartridge. The primer is placed as the second stick from the bottom in each hole.

Explosives.- Drift rounds in the ore are generally blasted with 27 per c balanced dynamite or 40 per cent strength ammonia dynamite in harder ground. Fo per cent strength gelatin dynamite is used in raises. The cut holes of rounds i hard rock are blasted with 80 per cent strength gelatin dynamite, if necessary, the other holes with 40 per cent strength of the same explosive. Cut holes of l rounds are loaded with 80 per cent gelatin. The correct amount of powder for ea round is issued to the miners upon the order of the shift bosses.

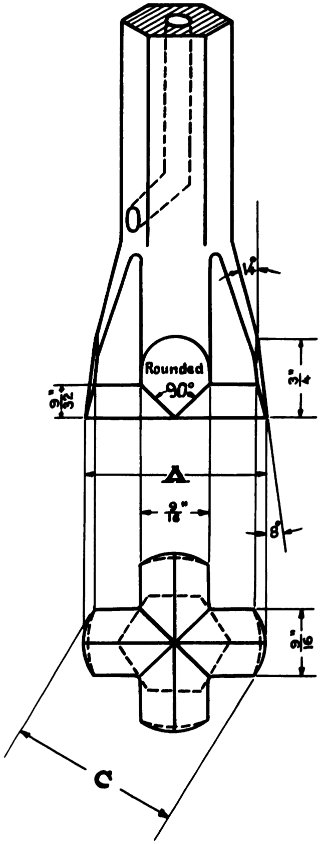
Detonators.- All blasting in raises over 30 feet high and in shafts is d with electric delay detonators. Some drift rounds are also blasted in this mann

Flat shoulder



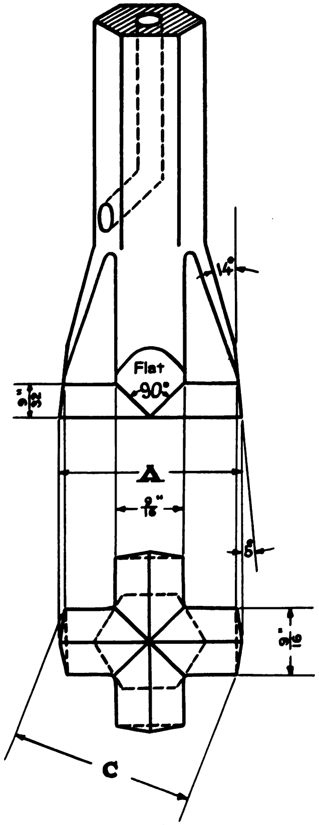
Length	Size of bit	
	A	C
2'- 6"	1 11/16"	1 11.26/16"
3'- 9"	1 10/16"	1 10.32/16"
5'- 0"	1 9/16"	1 9.38/16"
6'- 3"	1 8/16"	1 8.43/16"
7'- 6"	1 7/16"	1 7.52/16"
8'- 9"	1 6/16"	1 6.60/16"

Round shoulder



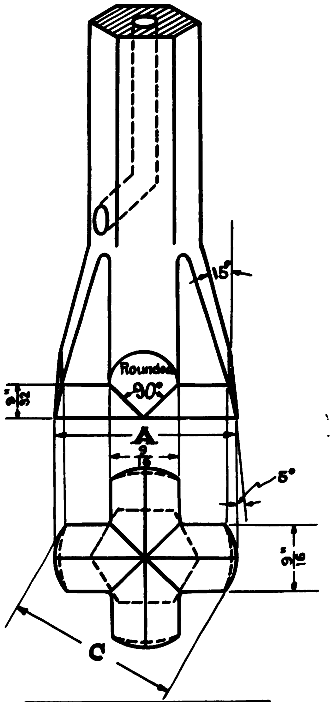
Length	Size of bit	
	A	C
2'- 6"	1 11/16"	1 9.74/16"
3'- 9"	1 10/16"	1 8.74/16"
5'- 0"	1 9/16"	1 7.74/16"
6'- 3"	1 8/16"	1 6.74/16"
7'- 6"	1 7/16"	1 5.74/16"
8'- 9"	1 6/16"	1 4.74/16"

Flat shoulder



Length	Size of bit	
	A	C
2'- 6"	1 11/16"	1 11.72/16"
3'- 9"	1 10/16"	1 10.77/16"
5'- 0"	1 9/16"	1 9.84/16"
6'- 3"	1 8/16"	1 8.90/16"
7'- 6"	1 7/16"	1 7.97/16"
8'- 9"	1 6/16"	1 7.05/16"

Round shoulder – Original



Length	Size of bit	
	A	C
2'-6"	1 11/16"	1 10.21/16"
3'-9"	1 10/16"	1 9.21/16"
5'-0"	1 9/16"	1 8.21/16"
6'-3"	1 8/16"	1 7.21/16"
7'-6"	1 7/16"	1 6.21/16"
8'-9"	1 6/16"	1 5.21/16"

FIGURE 3. Double taper drill bits

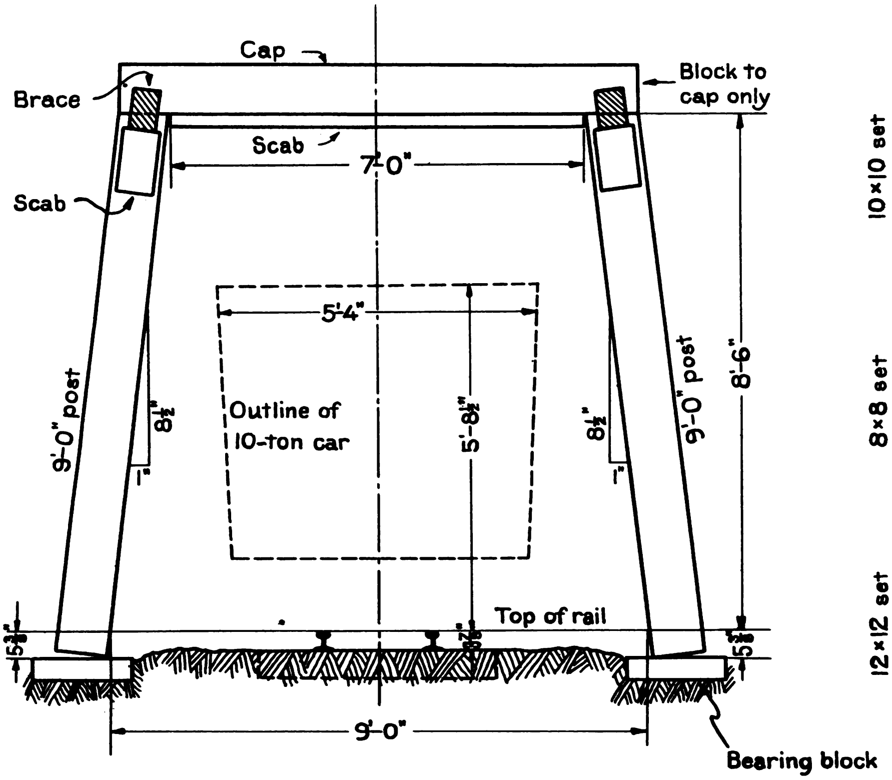
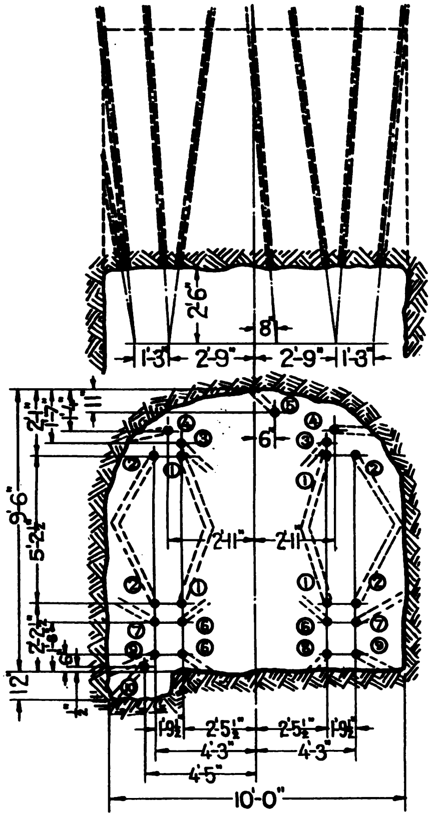


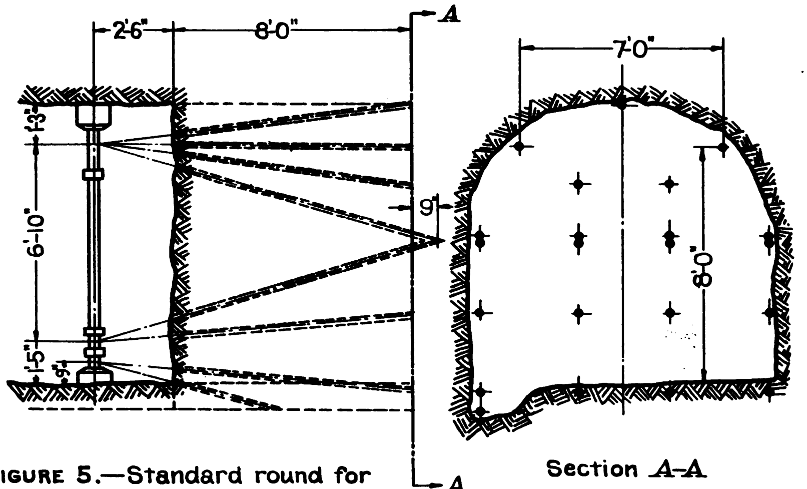
FIGURE 4.—5th level standard drift set

	LIST OF TIMBER FOR 1 SET				
	PIECES		NO.	DESCRIPTION	BO.FT.
10×10 set	2	Posts	39	Batter 1 End 10x10x9-0 As Shown	150.0
	1	Cap	42	10x10x8-8 Sq Ends	72.2
	1	Scab	128	2x10x7-0 " "	11.7
	2	Scabs	44	2x8x1-0 " "	2.7
	2	Braces	63	6x8x4-2 " "	33.3
	2	Bearing Blocks	45	4x12x1-8 " "	13.3
				TOTAL BOARD FT.	283.2
8×8 set	2	Posts	46	Batter 1 End 8x8x9-0 As Shown	96.0
	1	Cap	49	8x8x8-4 Sq. Ends	44.3
	1	Scab	11	2x8x7-0 " "	9.3
	2	Scabs	44	2x8x1-0 " "	2.7
	2	Braces	62	6x8x4-4 " "	34.6
	2	Bearing Blocks	45	4x12x1-8 " "	13.3
				TOTAL BOARD FT.	200.2
12×12 set	2	Posts	52	Batter 1 End 12x12x9-0 As Shown	216.0
	1	Cap	55	12x12x9-0 Sq. Ends	108.0
	1	Scab	27	2x12x7-0 " "	14.0
	2	Scabs	44	2x8x1-0 " "	2.7
	2	Braces	61	6x8x4-0 " "	32.0
	2	Bearing Blocks	45	4x12x1-8 " "	13.3
				TOTAL BOARD FT.	386.0

Blasting chart							
Holes	Order of firing	Kind of powder and amount per hole					
①	First						
②	Second						
③	Third						
④	Fourth						
⑤	Fifth						
⑥	Sixth						
⑦	Seventh						
⑧	Eighth						
⑨	Ninth						
⑩	Tenth						



183 feet of drilling



Section A-A

FIGURE 5.—Standard round for 9' 6"×10' drift, 8' 0" break (21 holes + 1 short hole for ditch)

All blasting in pillars on the undercutting level, on grizzlies, and in hung-up raises is by means of instantaneous electric detonators.

All blasts, except block holes, are detonated by No. 8 straight caps.

Fuse with No. 8 detonators are used in most drift work.

The caps of all electric detonators used in rounds are No. 8 in strength.

Rounds are blasted from the mine circuit of 250 volts. A special circuit-breaker switch for electric blasting is used, which breaks the circuit automatically and almost instantaneously after being applied in the blasting circuit. Since this switch was adopted no trouble from side-spitting has occurred.

Stemming.- Stemming in cartridge form is used in all rounds. The cartridges are made by packing a powder box full of paper shells with open ends upward. The shells are then filled with fine sand tailings and the ends of the shells closed by hand.

Drifts

The contracts for driving drifts are generally divided into two parts: (1) Breaking and removing the rock; (2) timbering, including squaring out the section. Occasionally, where the ground requires timbering close to the face, breaking and timbering are included in the same contract.

Three general types of drifts are run: (1) Motor haulage, 10 by 10 feet in cross section; (2) supply, 6 by 8 feet in cross section; and (3) grizzly and undercutting, 4 by 6 feet in cross section.

Motor Haulage Drifts.- The motor haulage drifts are generally broken 10 by 10 feet in cross section in the timbered sections, and 8 by 9 feet in untimbered sections. The timbered sets under the stopes are 8 feet 6 inches from the rail to the bottom of the cap (fig. 4).

The depth of the round broken depends upon the cycle of operations desired and the condition of the ground in regard to timber support.

In ground where falls from the back may occur before the round is mucked out, protection is afforded the workmen by means of booms or stringers. After the round is shot the cap and top lagging for the next set is put in place and rested on two booms. The stringers are supported under the cap of the last set stood by heavy bars of iron bent to fit the stringers; the two ends are turned to hook over the top of the cap. The rear ends of the stringers are blocked down under the cap of the second set from the face.

After the broken rock is removed, the posts of the sets are stood and the cap is lowered on the posts by knocking out the blocking over the rear end of the stringers. Figure 5 shows a standard 8-foot drift round.

Grizzly Drifts.- Raises are run from the haulage drift to reach the botto of the grizzly drifts at intervals of 14 or 21 feet before those entries are extended. Most of the ore broken by blasting falls into the raises. The rest is shoveled into them.

Undercutting Drifts.- After the finger raises are put up the undercutting drifts are run. The tops of the raises are on 14-foot and 18 2/3-foot centers. The undercutting drifts are advanced by the same method as on the grizzly level. The costs are less, as the raises for receiving the broken ore are closer togethe and the undercutting drifts are not timbered except for an occasional stull.

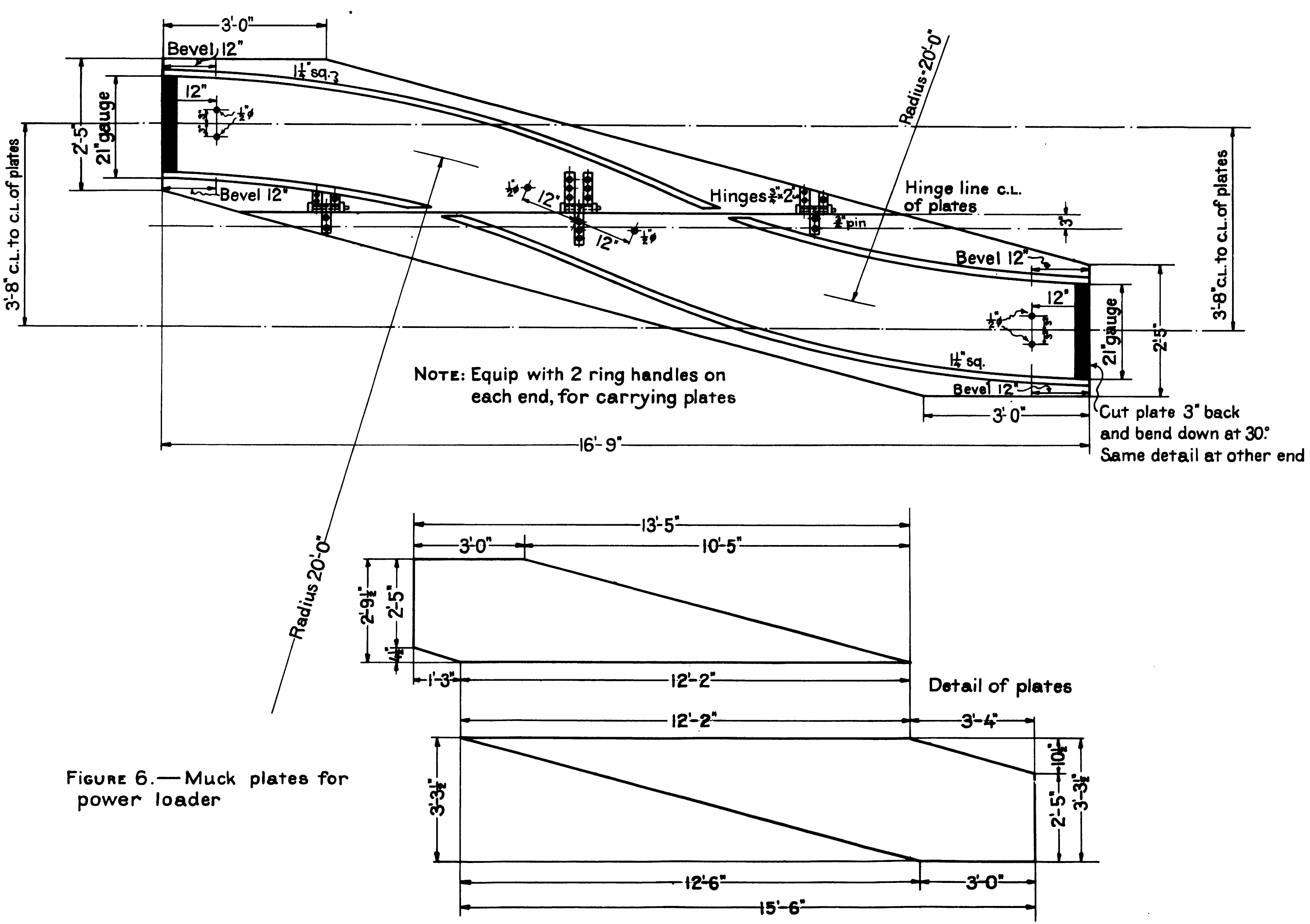
Use of Mechanical Loader.- The rate of advance of any single heading is generally dependent upon the time required for the removal of the rock broken by blasting. A very material saving in time was made by using a mechanical loader while advancing the 14th-level main heading.

The shovel used in this work has a capacity of 30 tons per hour. It consists of a frame similar to that of a mine locomotive supporting a conveyor and scoop, and is electrically driven. Unlike many other types of mechanical loaders it does not clamp to the rail, but it travels back and forth on the track during operation. As the shovel is moved forward the scoop digs into the pile of broken material. As it moves back again the scoop is raised, and the rock slides to a conveyor belt which discharges into the mine car. Power is transmitted through sprocket chains and clutches. The movement of thrusting the scoop into the muck pile is accomplished by driving the whole machine forward on the tracks, which avoids the complicated differential motion necessary in other small power loaders

Organization of the drifting work on this level had to be completely changed to make the best use of the mechanical loader, and as a direct result it was some time before an acutal gain in daily advance was obtained. However, between February 1, 1928, and September 30, 1928, 2,343 feet was driven in the mai heading, having an average rectangular cross section of 10 1/2 feet in width and 9 1/2 feet in height. Approximately 9 tons of rock was broken per foot advance. The actual daily advance was 11.3 feet, as compared to 7.3 feet by hand-shoveling which constituted a gain of 54.8 per cent. Delays during working periods for emergency repairs to the shovel averaged less than 15 minutes per shift.

Previous to installing the loader, four men were employed on the shoveli crew, two mucking and two handling the cars. Progress was slow and discouraging the muckers because of water in the heading which was being driven down-grade. This work, so distasteful to the men, was eliminated by the use of the mechanica loader. The best crew for the loader consisted of three men, one man handling t main controls, one man tending empty and loaded cars, and one man dividing his t between handling the dipper controls and tramming.

The drifts were wide enough to permit double track for 1-ton cars, which insured a good supply of cars to the loader. A portable switch or crossover (fi 6) was used, which reduced the delay while cars were being trammed to and from t loader.



Note: Equip with 2 ring handles on each end, for carrying plates

Cut plate 3" back and bend down at 30°. Same detail at other end

Figure 6.—Muck plates for power loader

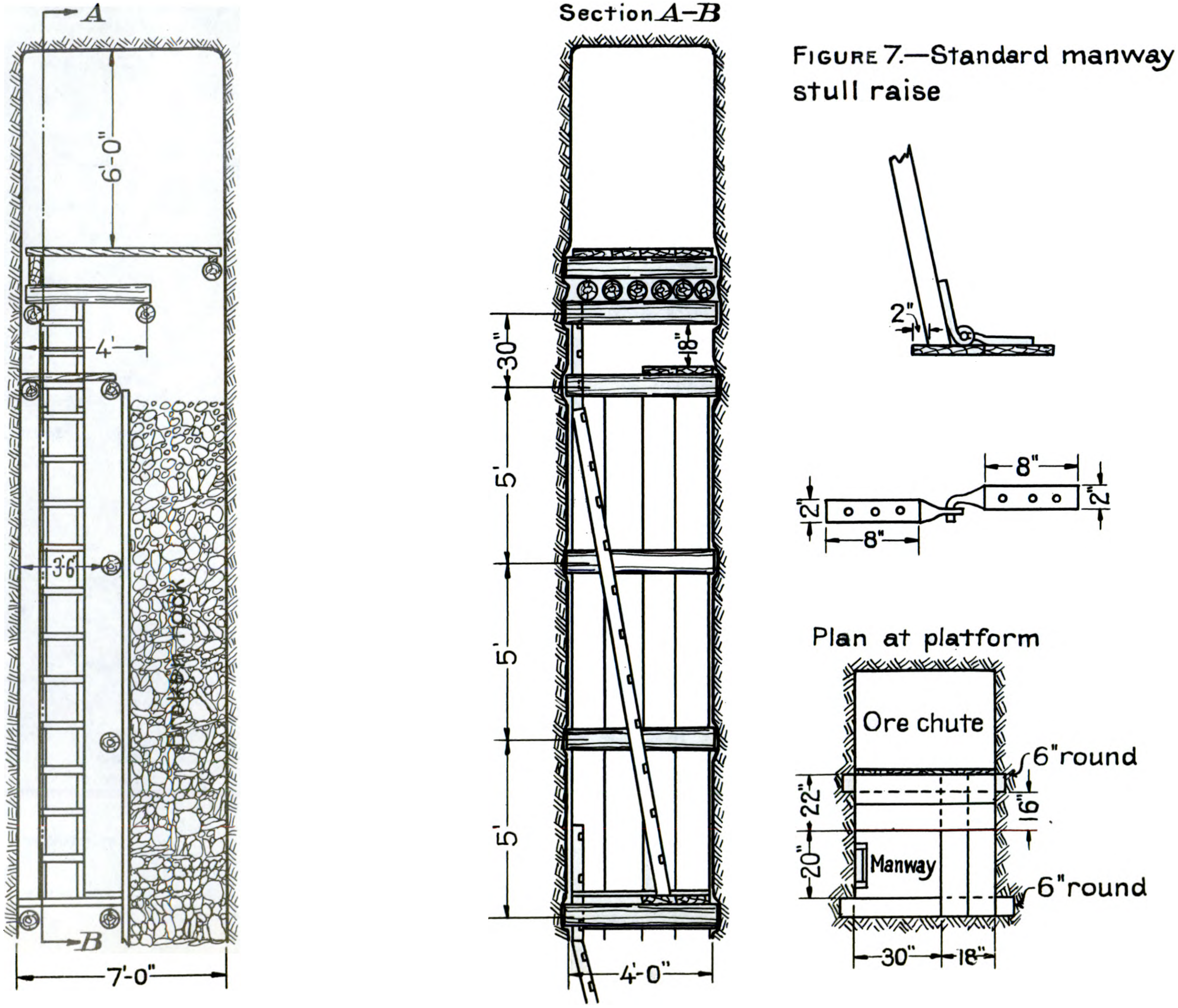


FIGURE 7.—Standard manway stull raise

Ordinarily two complete cycles were made per day--that is, two rounds were drilled and shoveled out in each 24 hours. The following tabulation gives the average total time spent on each operation for the 12 hours required to complete a cycle:

Operation	Time (hrs. and min.)
Setting up	0.45
Drilling	3.30
Loading and firing.	.45
Tearing down	.15
Waiting for smoke to clear	.45
Mucking	6.00
Total	12.00

The relative distance mucked per man shift in a section 1,788 feet long where hand-shoveling was used and in 2,343 feet run with the loader was 1.65 and 1.80 feet, respectively. Although when the loader was used a slight increase in feet mucked per man-shift was apparent; the actual driving efficiency dropped. This was due primarily to the necessity of speeding up operations and consequently resulted in the unavoidable crowding of men in the face. Although the result was a drop in the efficiency for this size of heading the decrease was more than offset by the increase in daily advance. This crowding in the face also increased the cost per foot of advance. The loader proved successful, as the work was done in 114 less operating days than would have been necessary had all hand labor been employed.

This operation was planned for rapid progress. Lower mucking costs may be obtainable by reducing the number of men and adapting the organization to the most efficient mucking rate.

Raises

From the experience in the district it has been found that ore transfer raises run at an angle of 70° are the most satisfactory, as at this inclination the ore neither builds up on the bottom side of the raise nor packs in the chutes.

Raises 4 by 6 feet in cross section and 50 feet high for ore chutes are run either vertically or on an incline of 75° from the haulage drift to the grizzly level and are not ordinarily timbered. The inclination used fits the spacing of the grizzlies. Where the ground will not stand the raises are cribbed. One man drills and blasts a round in these raises in a shift or less and generally breaks about 4 feet. No water or air pipe lines are taken up these raises. A round generally consists of 9 holes, which are each loaded with four or five cartridges of 40 per cent gelatin dynamite.

Figure 7 shows the details of the pilot raise for shrinkage stopes.

Pilot raises are used for making shaft connections. After the connectio are made the raises are enlarged to full shaft size, beginning at the point of connection and carrying the timbering downward.

For distances of over 200 feet, raises are run with three compartments. The size of the compartments and the timbering details are shown in Figure 8. O compartment is used for holding the broken rock, the middle one is a manway, and the third compartment is used for a light cage to handle men and supplies used i the raise. Stations are cut every 100 feet. As soon as a raise has progressed feet above a substation, the sheave for the hoist is raised and hoisting is done to that level.

To reduce the excessive weight of broken rock at the bottom of the chute of such a long raise, rock is transferred through auxiliary chutes spaced 100 fe apart.

All raise rounds are blasted electrically, using five delays. Before lo ing the holes the miner doing the work obtains the only key to the lock of the blasting switch. After the round is loaded the man in charge connects the lead wires to the blasting line 25 feet from the bulkhead, where a second man is stationed. These men then descend to an interrupter switch at the bottom of the raise. After the approaches to the raise are guarded the interrupter switch is closed and the round blasted through the main blasting switch.

The back of the raise is never more than 12 feet above the timber. Afte a round is blasted a 5-foot set is put in place; on top of this a blasting set i installed, and then the next round is drilled. The broken rock in the raise is kept within 6 feet of the bulkhead. Just before blasting, enough rock is drawn from the chute to provide space for the material broken by the blast.

Shafts

The Liverpool shaft was sunk first, and development of the lower levels the mine proceeded from it. The other shafts are deepened by first driving pilo raises from the lower levels.

The main shafts of the Humboldt mine are shown in Figure 2. No. 6 shaft contains two 5 by 6 foot hoisting compartments and one 5 by 5 foot manway. All for the mill is now hoisted from the 5th level, but plans are under way for dee ing the shaft by raising from the 14th level.

The Joy service shaft has one 7 by 13½ foot cage compartment and one 3½ 13½ foot compartment divided for the cage counterweight and for pipes, wire con and a ladderway. The cage is large enough to take a truckload of timber and wil carry 60 men comfortably.

The Liverpool shaft has two compartments above the 14th level and three below.

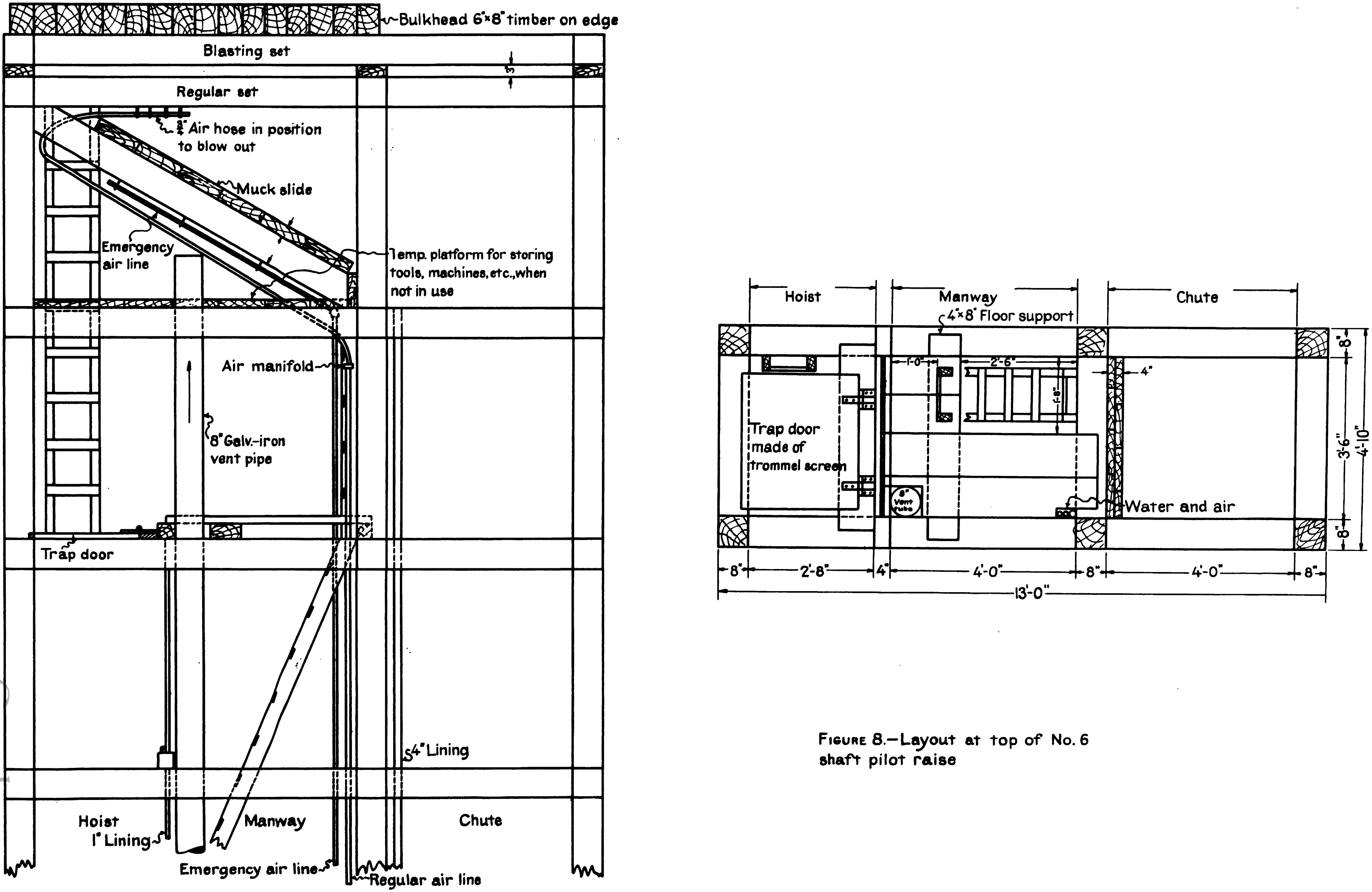


Figure 8.—Layout at top of No. 6 shaft pilot raise

EARLY MINING METHODS

Early mining methods in the Morenci district were very simple. After nearly 10 years of mining, during which the Longfellow mine had been worked over an area of roughly 500 by 600 feet, no timber, ore cars, hoists, or power in any form had been used. Open stopes had backs 80 to 100 feet wide supported only by pillars of ore. These pillars had been thinned in times of financial stress, until the mine began to cave over large areas. Entrance to the mine was by means of gently sloping inclines from the hillside, and the broken ore was pushed up the inclines in wheelbarrows.

This was the situation faced by the Arizona Copper Co. in its first years. In addition, the caving had tied up the best of the remaining rich ore in the Longfellow mine. Square-set stoping now came into general use, as it had been in other Arizona camps for many years. Tunnels were driven to reach the ore at lower levels, and tramming in cars was introduced.

In 1906 it was reported that attempts were being made to introduce caving methods into the district at the Shannon mine and elsewhere.[3]

By 1910 the Metcalf, Morenci, and Coronado mines were being worked by many different methods - open cut, surface and underground glory holes, open stope, cut-and-fill, shrinkage, and overhand and underhand square-sets.

In addition, two caved stope methods were in use, top slicing and sublevel caving. The development work for both methods was the same. Crosscuts were driven through the ore on 30-foot centers. Raises from them at 25-foot intervals were put up to the capping. With the ore body thus defined stoping began. For top slicing a good timber mat was established, and the top of the ore body was leveled by square-setting two or three floors. Then top slicing proceeded downward by what is still the accepted modern practice. Headings were run from the raise to the edge of the ore, and crosscuts were made for the width of the panel. Floors 9 to 11 feet high were mined, retreating to the raise.

In the so-called "block-caving" method a sublevel was first driven in the ore 20 to 35 feet below the top of the ore body. The larger figure applied in ground which broke easily and regularly. The sublevel consisted of untimbered drifts and crosscuts on 20 to 35 foot centers. Each of the rectangular blocks thus laid out was divided into four parts by further drifting and crosscutting. Caving was started by drilling and blasting the pillars between the drifts and crosscuts and at the backs of openings, thus undercutting and weakening a block so that it fell and crushed of its own weight. The broken ore was extracted partly by shoveling from the edge of the caved block into the yet uncaved openings and partly by extraction raises put up under the caved area. When waste appeared drawing stopped.

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3 Woodbridge, Dwight E., "Mining, milling, and smelting in the Clifton-Morenci-Metcalf district": Eng. and Min. Jour., July 21, 1906, pp. 83-103.

In 1917 an inclined top-slicing method was in use in the Coronado mine of the Arizona Copper Co.[4] This method had the advantage of eliminating all shoveling of ore in the stopes (fig. 9). The broken ore was blasted down on the slopin floor of the stope from which it ran into a narrow shrinkage stope at the bottom of the trough. The ore from the shrinkage stope was drawn directly into cars through chutes on the motor haulage level. As a result of the substitution of in clined top slicing for flat top slicing, the tons of ore mined per man shift in top-slice stopes increased from 4.90 in 1916 to 11.20 in 1917. The total cost of labor and material decreased from $1.06 to $0.75 during the same period.

A variation of the same method of inclined top slicing was adopted in the Humboldt mine at Morenci in 1918.[5] At this mine the development work consisted o running sublevels at 55-foot vertical intervals and extending raises from the sub levels to the stopes. Each sublevel had a central drift through the ore body, with crosscuts at right angles to it at 50 or 60-foot intervals. From the latter untimbered raises not over 4 feet in diameter were put up every 15 feet. These lines of raises formed the center lines of the individual slices. In stoping, an inclined heading was driven 44 feet above the sublevel, starting at one end of th line of raises. This heading was timbered with sets 11 feet high and 2½ feet apart. As soon as the heading had gone far enough to intersect four raises stopi was begun. The ground was taken out in panels 10 feet wide and 50 or 60 feet lon corresponding to the distance between drifts on the sublevel. The floor of the panel was inclined 33° up on either side of the center heading, and the floor of the heading was funneled down to raises, so that ore blasted from the faces of th panels would run into one of the raises without any handling. When the heading had advanced to open another raise, which completed the panel, the timber was blasted and another started. Posts in the panels were 8-inch round timber, 10 f long, on 4-foot centers up the incline and 5-foot centers parallel to the headin These posts were not set normal to the floor of the panel but inclined toward the center heading at an angle of 16½ degrees from the vertical. Flooring consisted of 2 by 12 inch planks laid upon sills of round timber. The sills were so placed as to be readily picked up by the posts of the next lower slice. The ore was drawn from chutes on the sublevel and hand-trammed to motor-haulage pockets. Th substitution of inclined top slicing for flat top slicing resulted in increasing the tons mined per man shift from 5.4 in 1917 to 8.8 in 1919. The timber consum tion was 7.9 board feet per ton with flat top slicing and was 9.0 board feet wit inclined slicing. The total cost showed a reduction of 15 per cent in favor of the inclined method.

The next development at Morenci was a modern caving method--"The Morenci timbered slide caving system." From 1923 to 1926 this method was used for minin a large part of the Morenci ore.

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[4] Scotland, P. B., "The Coronado top-slicing method": Eng. and Min. Jour., April 7, 1917, pp. 103-561.
Scott, W. G., "Incline top-slicing method": Trans. Am. Inst. Min. Eng., 1918 pp. 59-305.

[5] Hodgson, J. P., and Kiddie, John, "The incline slicing method as applied to large ore bodies": Eng. and Min. Jour., May 27, 1922, pp. 113-914.

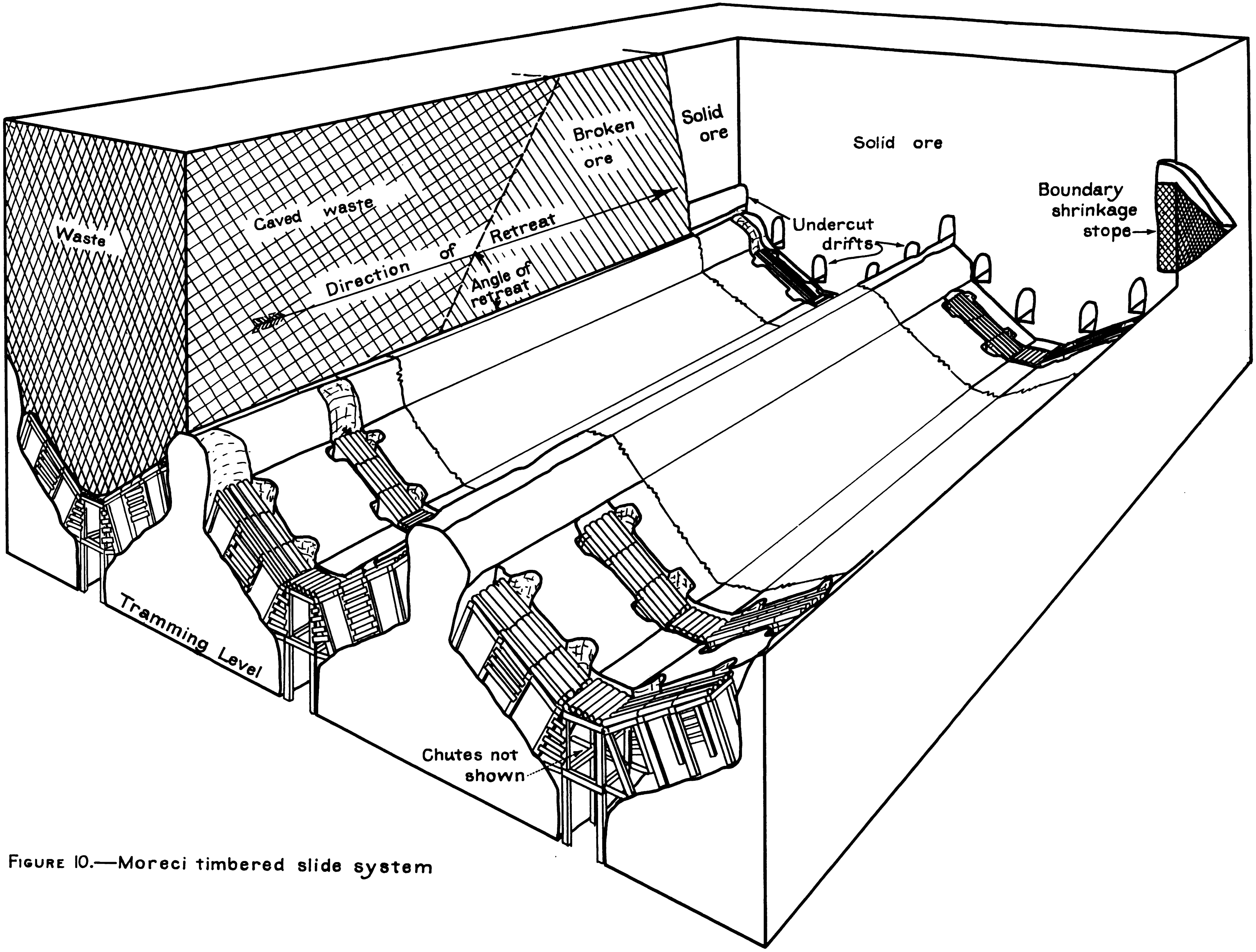
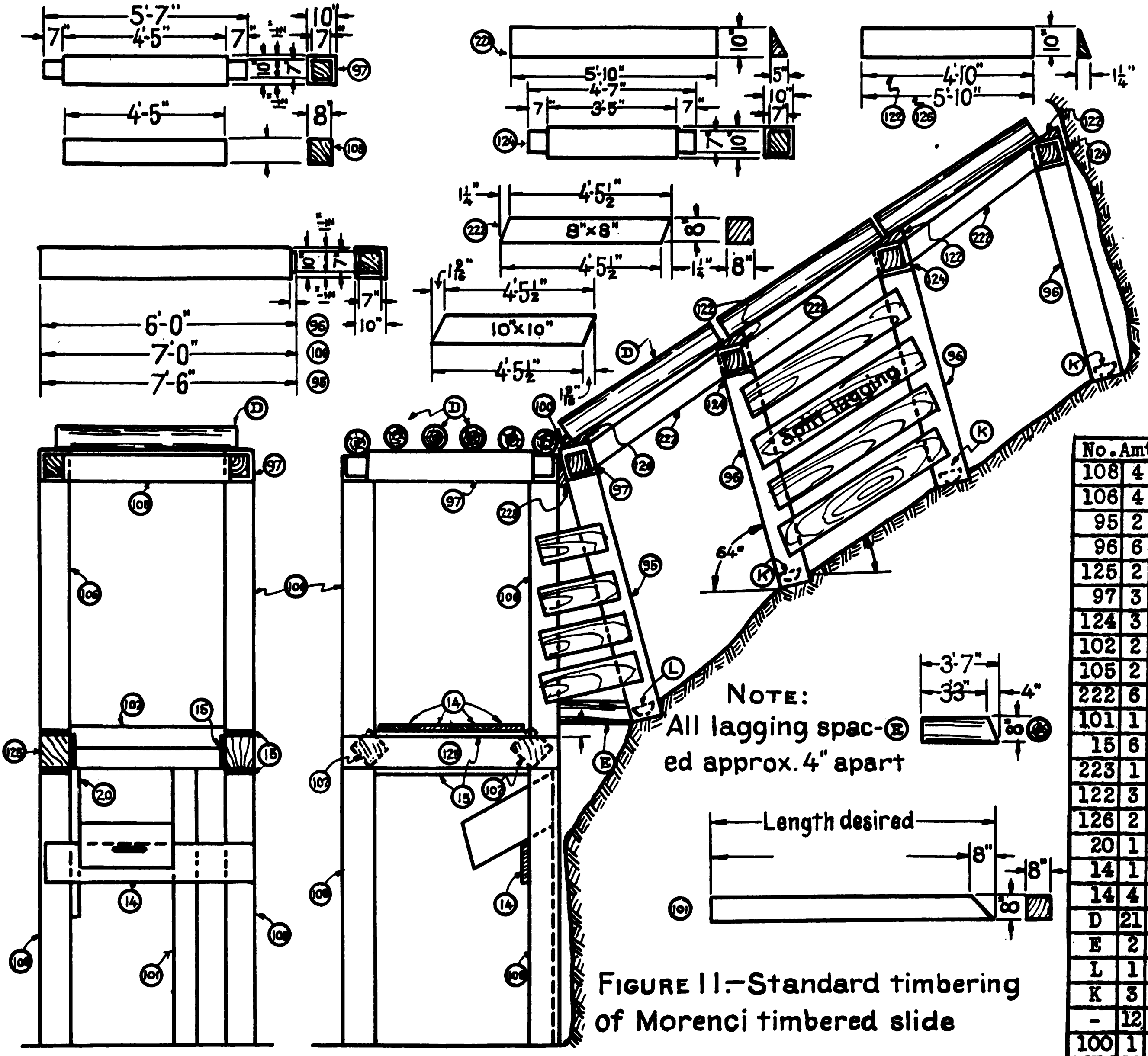


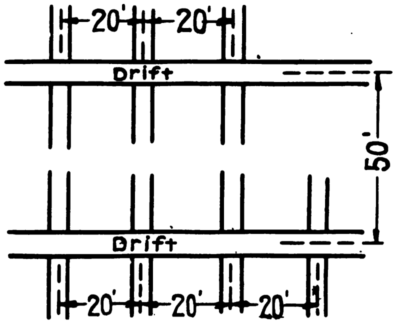
FIGURE 10.—Moreci timbered slide system



Note:
All lagging spaced approx. 4" apart

FIGURE 11.—Standard timbering of Morenci timbered slide

Plan
Morenci timber slide



TIMBER SCHEDULE

No.	Amt.	Name	Dimension	Description	Bo. Ft.
108	4	Post	10x10x7-6	Square Ends	250.0
106	4	Post	10x10x7-0	See Detail	233.3
95	2	Post	10x10x7-6	See Detail	125.0
96	6	Post	10x10x6-0	See Detail	300.0
125	2	Cap	10x10x5-10	Square Ends	97.2
97	3	Cap	10x10x5-7	See Detail	139.5
124	3	Cap	10x10x4-7	See Detail	114.6
102	2	Brace	10x10x4-2	Square Ends	69.4
105	2	Brace	8 x 8x4-5	Square Ends	47.0
222	6	Brace	8x8x4-6¾	See Detail	131.8
101	1	Post	8x8x L.D.	See Detail	40.2
15	6	Scab	2x10x4-2	Square Ends	41.4
223	1	Scab	3x10x5-10	See Detail	24.3
122	3	Scab	1¼x10x4-10	See Detail	42.2
126	2	Scab	1¼x10x3-10	See Detail	31.5
20	1	Scab	3x10x4-0	Square Ends	10.0
14	1	Scab	2x12x5-10	Square Ends	11.6
14	4	Plank	2x12x5-10	Square Ends	46.6
D	21	Lags	8"rndx5-0	Square Ends	439.7
E	2	Brace	8"rnd.x3-7	See Detail	16.7
L	1	Brace	Split-4-2	Square Ends	4.9
K	3	Brace	Split-3-2	Square Ends	11.2
-	12	Split Lags 5' long		----	64.3
100	1	Scab	2x4x5-10	Square Ends	3.9
				Total Board feet	2,291.

In this system of mining a block or panel 150 feet wide is developed by three parallel hand-tramming drifts run on 50-foot centers (fig. 10). In these drifts every fourth drift set carries a pony set on top of it. From each side of the pony set an incline is driven at right angles to the tramming drift below and is sloped upward at an angle of 33°. Pony sets and inclines are timbered (fig. 11). The inclines do not connect with the corresponding inclines from the next tramming drifts but are stopped so as to leave a 10-foot pillar between. This completes the development work proper.

The next step is the first stage of undercutting and consists of driving small untimbered drifts or "dog holes" parallel to the tramming drifts and just above the inclines or slides. These drifts are spaced on 10-foot centers. Where they cross over the inclines, the floor is cut down so that broken ore from driving and undercutting will roll into the slide without shoveling. A shrinkage stope is run on one or both sides of the panel and across the ends. Where the panel is bounded by caved ground this is not necessary. The shrinkage stope varies from 5 to 10 feet in width and is carried within 5 or 10 feet of the top of the ore. It is mined with a stepped back like the back of a rill stope, in order that entrance may be had at the low point from an undercut drift or the end of a timbered incline. This makes it unnecessary to have cribbed raises for entrance to the stope or manways from the level above.

Actual production begins with the second stage of the undercutting process, which consists of blasting down the pillars between the undercut drifts. At the same time holes 8 feet long are drilled and blasted in the backs of the drifts. Caving is started at one end of the panel and progresses to the other end. Undercutting and drawing are controlled so that the "angle of retreat," which is the angle with the horizontal made by the theoretical plane separating ore and waste in the working stope, varies between 50 and 80°, depending upon weight developed in the stope.

PRESENT MINING SYSTEM

Development of Method

Before 1921 the Humboldt ore body had been mined principally by top slicing and square-setting, with a small proportion of drift caving. At the time of the general shutdown in that year the ore was left in an irregular condition, satisfactory for a continuation of the former methods but not convenient for starting production by caving.

During the years from 1915 to 1922 caving practice had been well established, and the principles were better understood. The reduction in operating cost to be expected from a caving method, although accompanied by a moderate reduction in the assay grade of the ore produced, indicated a good net gain in profits. For this reason it was decided in 1922, when production was resumed, to adopt a caving method for Morenci. At that time several mines were being worked by caving systems which varied considerably in operating details. The method introduced at Morenci, and from which the present Morenci undercut block-caving system has been developed, embodied the features that were believed to be best suited to the conditions there.

The immediate problem upon starting operations was to resume production as soon as possible. In order to do so, existing development and stope-preparation work was made use of, although it was designed for different mining methods. Various ingenious methods were improvised to take advantage of local conditions; thus the Morenci timbered slide was developed, which is still preferable in certain ground for lifts of less than 100 feet. For the larger blocks, modifications of several caving systems, standardized in some of the western mines, were introduced.

Principles of Mining by the Undercut Block Caving Method

Extracting ore by caving consists of two operations: (1) Undercutting, and (2) drawing off the ore through openings uniformly spaced horizontally below the caving block. By undercutting is meant the blasting and partial or complete removal of a layer of ore across the base of the column to be mined, so as to permit the ore above to cave and break up by gravity. In some districts the ore has at times been broken by vertical shrinkage rooms with pillars between.

The method of breaking up a layer of ore sufficiently thick to cause satisfactory caving is a matter of convenience or economy. For this reason caving practice is not uniform at different mines.

Undercutting is supplemented by breaking loose the caving block from surrounding rock or ore by cutting or weakening it along its boundaries, so as to assist the caving action and confine it to the block mined.

For perfect drawing the draw points should be closely and regularly spaced; but a compromise must be made between close spacing, which increases ore recovery and reduces dilution, and wider spacing, with resulting economy in preparation costs and in operating repairs. The character of the ground controls drawing practice, which varies in details but not in theory.

Dilution is caused by the irregular movement of ore and capping toward the drawing points with the formation of pipes of waste which may reach a raise far in advance of the top boundary of the ore. Dilution may also be caused by the genera infiltration of fine capping down through coarsely broken ore. In all caving, moreover, there is a gradual mixing of ore and capping as the ore is drawn down. If properly drawn a large part of the ore comes to the chutes clean and before appreciable dilution occurs. From that point the proportion of waste increases and that of ore decreases.

The end point of drawing is set by the value of the ore and the profit per ton or per year at that content of copper yielding the best financial returns. If the plant capacity is large, drawing may continue until near the point of no profi

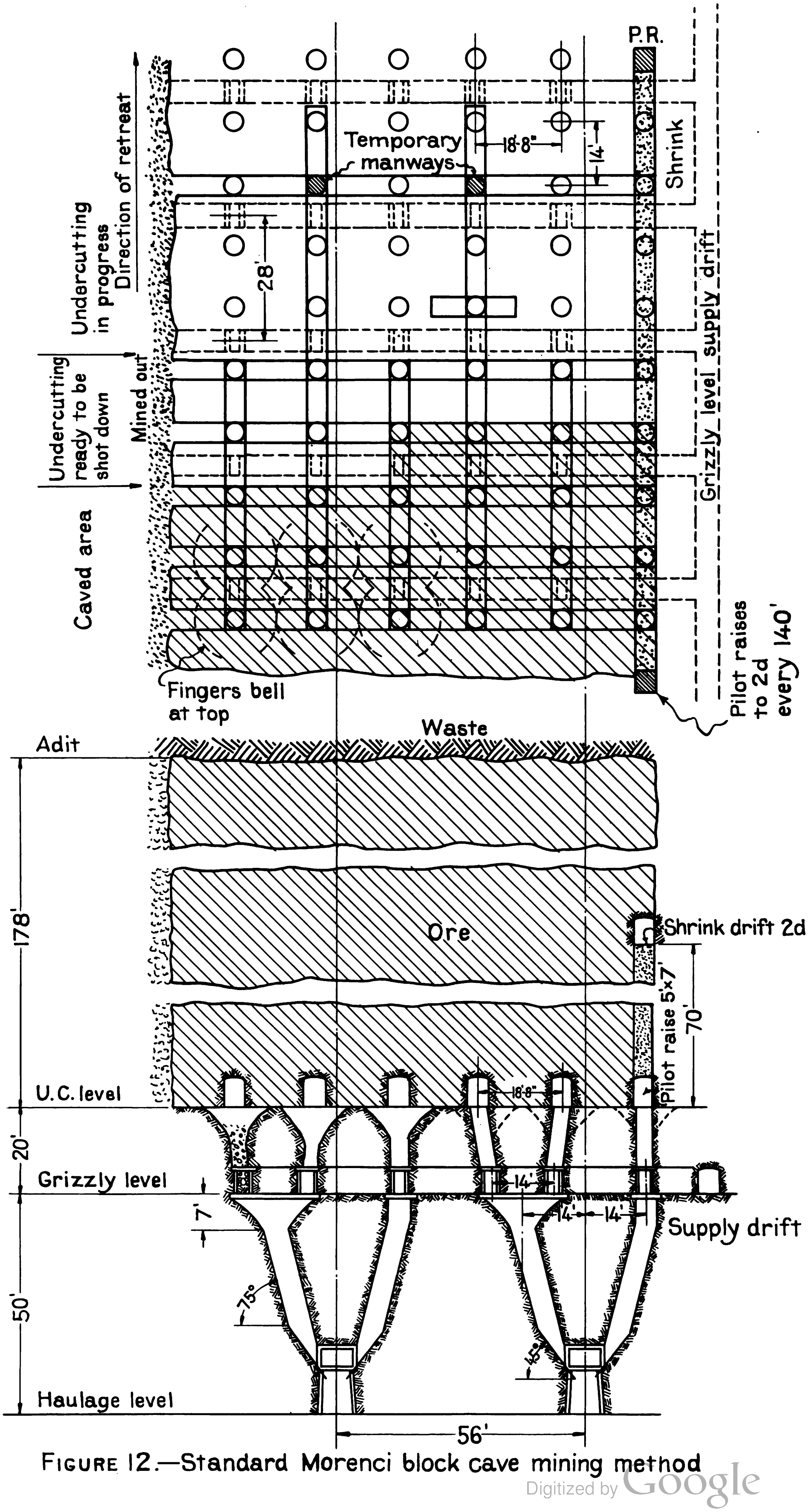


FIGURE 12.—Standard Morenci block cave mining method

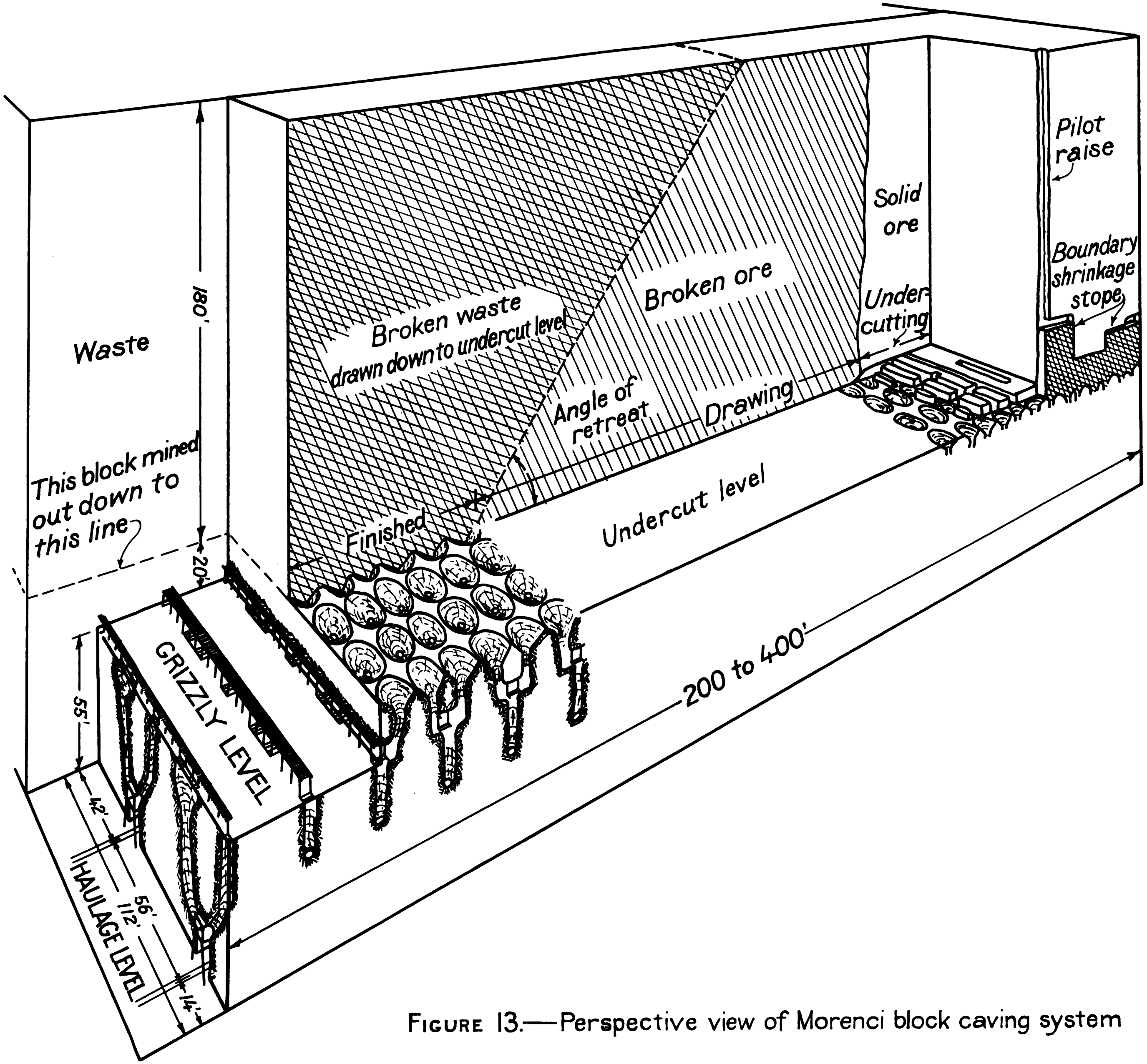


FIGURE 13.—Perspective view of Morenci block caving system

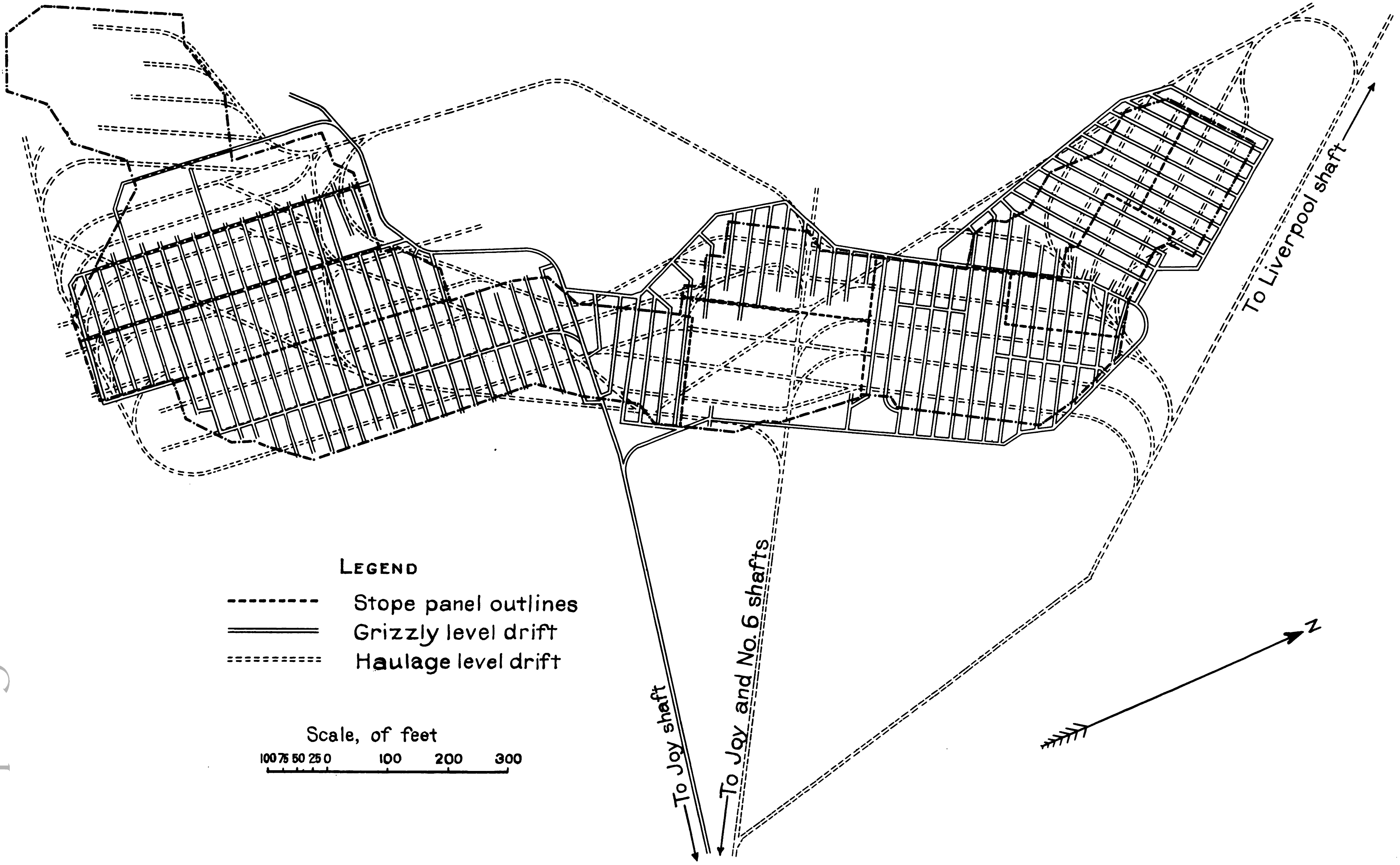
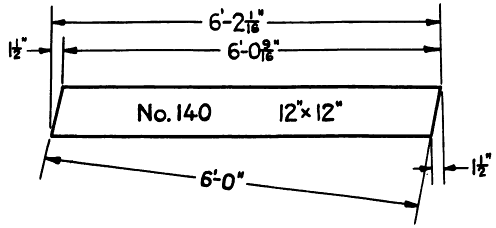


FIGURE 14.—Plan of grizzly and haulage levels, Humboldt mine



TIMBER SCHEDULE		
No.	Amt.	DESCRIPTION
321	8	12x12x3-6 Sq.Ends (2 invisible underpieces)
324	6	12x12x4-0 " "
322	12	12x12x5-0 " "
141	4	12x12x5-6 " "
68	8	12x12x6-0 " "
140	8	12x12x6-2-1/16 Beveled, See Detail
323	2	12x12x8-0 Sq. Ends
114	8	12x12x14-0 Sq. Ends
103	8	10x10x3-6 Sq. Ends
345	8	3x12x20-15/16 For 50 Rails
30	12	3x12x5-6 Sq. Ends
212	4	3x12x5-0 " "
12	4	2x12x12 Sq. Ends
16	8	2x12x1-9 " "
273	4	2x12x2-6 " "
29	4	2x12x3-6 " "
31	4	2x12x4-0 " "

Below are listed pieces not shown on this drawing but which may be required in abnormal conditions to substitute for certain pieces listed in above timber schedule.

344	-	10x10x5'-0" Sq. Ends
113	-	10x10x5'-6" " "
57	-	12x12x12'-0" " "
268	-	12x12x7'-0" " "
325	-	12x12x10'-0" " "

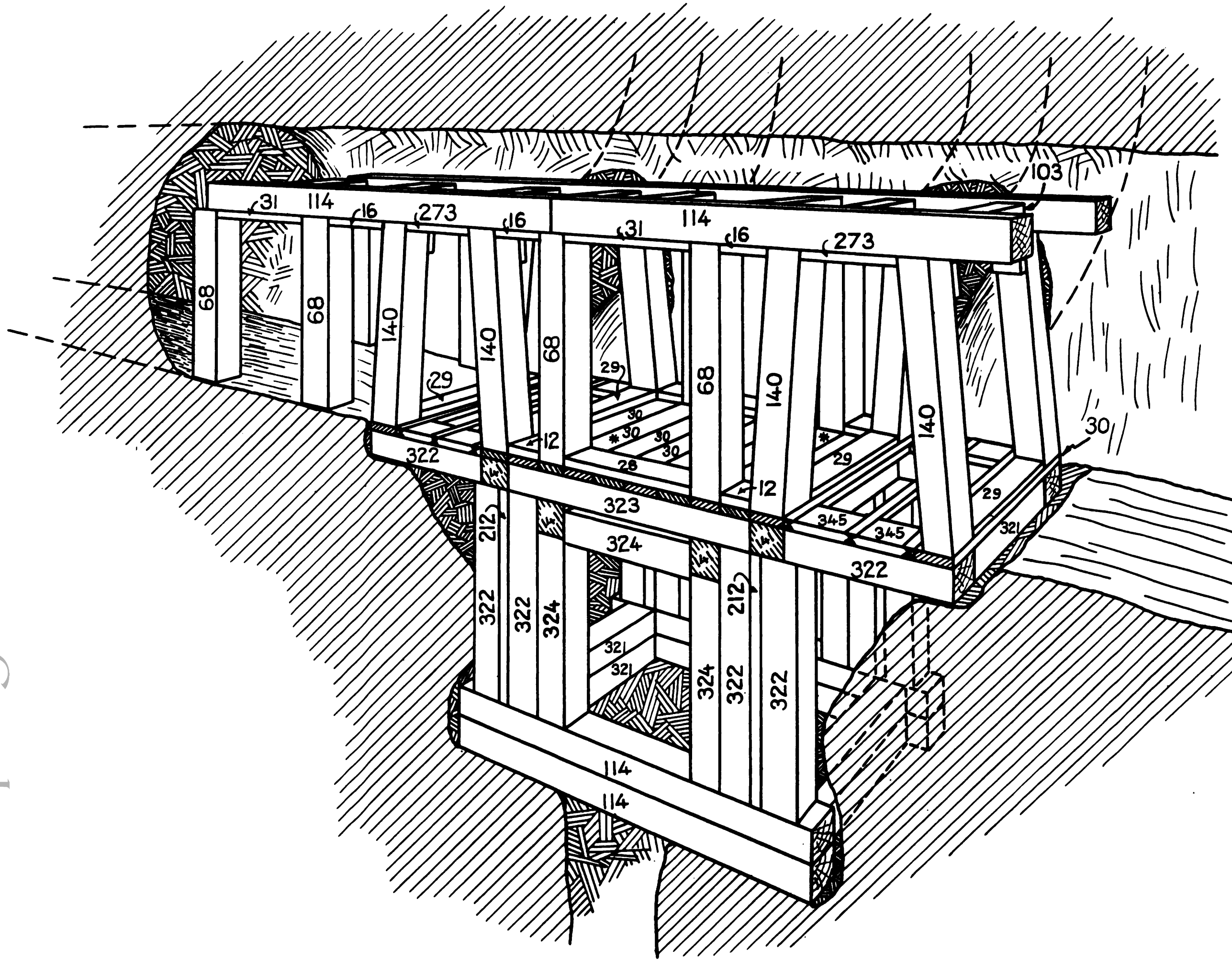


FIGURE 15.—Grizzly station

If not, to obtain adequate profits it must be stopped soon after dilution appears. The system is elastic in that by the sacrifice, for example, of 15 or 20 per cent of the ore its grade can be maintained at very nearly its original metal content.

Caving is characterized by intensive production from small areas and the repetition of a number of simple operations, which leads to the thorough study and complete standardizing of nearly all operations.

Stoping System

The Morenci caving method uses an undercutting level and a grizzly level 20 feet below it. The latter is connected by modified branch raises with a haulage level 50 feet lower. The raises carry the ore passed through the grizzlies to the haulage chutes (figs. 12 and 13).

For reasons of economy and speed in starting production the haulage level already established and in use when caving operations began was extended to gather the ore from the mining area and to deliver it to the shaft. A grizzly level was driven only 50 feet above the haulage level to reach as much ore as possible and still have reasonable storage capacity in the haulage raises. It was connected with the Joy shaft to serve as a main line of communication for men and supplies to the mining area. By this arrangement, haulage on the level below was relieved of practically all interference from the passage of men and supplies.

The elevation of the grizzly level determined the height of draw in the first lift. The ore extended upward to irregular heights above the grizzly level. Its upper limit was the assay boundary between the ore and capping or occasionally the bases of old square-set or top slice stopes. Only incomplete records were available of the ore left about the old stopes.

For mining, the ore body is divided into stoping panels 112 feet wide, which extend more or less on the strike of the ore from one limit to the other. As mining proceeds from one end of the panel it can be followed by another stoping operation in an adjoining parallel panel after enough time has elapsed to allow the caved waste capping to settle in the finished stope (fig. 14).

Each panel delivers its ore to two haulage drifts, 56 feet apart, which ru beneath and parallel with the panel. Chute raises from the haulage drifts to the grizzly drifts are driven from the former at 28-foot intervals. The grizzly drift are driven at right angles to the haulage drifts and across the stope panels. The are 28 feet apart also, and each is directly over and connects with a line of raises from the haulage chutes.

As originally laid out, the grizzlies were set in the drifts 14 feet apart. At present they are symmetrically but unevenly spaced to suit the finger raises. The new arrangement brings the centers of certain pairs of grizzlies 14 feet apart, in which case the top of the haulage raise is widened along the grizzly drift by cutting a slot below the drift to collect the ore from each. This avoids branchin the raise, which leaves too weak a pillar between the branches, and has proved to be a stronger and cheaper arrangement (fig. 15).

Two short finger raises are connected with the grizzlies by openings 4 by 6 feet in section cut in the sides of the drifts. On the undercut level they are belled out until they meet. Below this they are widened as much as is safe in order to permit large bowlders to be drawn well down for convenience in breaking.

After the necessary finger raises are driven shrinkage stopes 6 feet wide are carried up on the block boundaries high enough to cut the ore free from adjoining ground and to guide the line of shearing to the top of the ore in the desired plane. The excess of ore broken in shrinking is drawn out through the finger raises.

The shrinkage stopes are started by first driving pilot raises 140 feet apart. They connect with an old intermediate mine level which is situated at a convenient height, 90 feet above the grizzlies. These connections serve to ventilate the pilot raises and shrinkage stopes and also are used as entries for men and supplies.

The shrinkage stopes in soft ground are usually carried up 35 feet. At this elevation it has generally been found possible to leave intact sections 50 to 60 feet in length, situated midway between the pilot raises, and to continue shrinking upward only in sections 40 to 50 feet on each side of the pilot raises. This results in a saw-tooth arrangement of broken and unbroken ground. These shrinkage sections reach the elevation of the intermediate level, 70 feet above the undercut, and if in the opinion of the operating staff it is necessary to weaken the boundary still further they may be driven higher.

In beginning the undercut, which follows the boundary shrinkage stopes, one or two drifts on the undercut level are run lengthwise with the panel across several grizzly lines for convenient access to the undercutting drifts. Undercutting drifts 4 by 6 feet in section, locally termed "dog holes," connect the tops of finger raises both laterally and longitudinally, thus forming a gridiron of drifts and leaving rectangular pillars between, about 10 by 14 feet in plan. Certain advance fingers are designed for handling supplies for the undercut sectio and for ladderways from the grizzly level, thus insuring a safe horizontal retreat for the men on the undercut level.

The checkerboard is developed, beginning at the area already caved, by connecting one or two rows of fingers across the panel and proceeding in the direction of the stope retreat.

The backs of all "dog holes" are drilled to a depth of 6 feet in fan-shape rings of stoper holes at 4-foot intervals. These holes are fired simultaneously with the pillar shots. The extra cut assists in starting the caving action and with the open finger raises provides room for the expansion of the broken ore.

The pillars are usually blasted down 2 or 3 at a time, beginning in the heaviest ground and progressing across the stope. Blasting is done with No. 8 instantaneous electric detonators, which have eliminated difficulties arising from cut-out holes as well as the hazards from loose powder in the ore drawn from the stope.

The advantages of the checkerboard system of undercut are: (1) The safety of the workmen who work in low drifts. There is no danger of drilling into a missed hole, for the second half of a pillar is drilled before the first half is shot. (2) The ventilation is good. Fresh air is drawn from the grizzly level through open fingers and is carried up through broken ground in the boundary shrinks to a primary exhaust air course on the intermediate supply level. (3) The method is elastic since the heavy ground may be shot out first, regardless of its location in the panel. (4) Immediate inspection of results is made possible by shooting only a small number of pillars at one time.

In opening a new panel 112 feet wide it is necessary to undercut through a length varying from 84 to 196 feet to break up arches or cantilevers and to start caving freely, even though the new panel may lie against the caved waste filling in a completed section.

The sequence of preparatory operations, based mainly on convenience in disposing of the ore broken in stope preparation, is as follows:

(1) Haulage drifts.
(2) Haulage raises.
(3) Grizzly drifts (a service drift on this level may be driven in advance of haulage raises).
(4) Grizzlies.
(5) Finger raises.
(6) Shrinkage stopes.
(7) Undercutting.

All these operations are carried on at the same time, each process being sufficiently in advance of that following it to avoid interference and provide convenient means for getting the ore to the haulage cars through haulage and finger raises.

In mining a block of ore 150 feet high, including the necessary development work and boundary shrinkage stopes on one side and two ends of the block, the amount of ore actually drilled and blasted is less than 10 per cent of the total ore removed.

After undercutting, drawing begins; the work is done slowly at first until the ore starts to cave freely, and afterward at the rate that best suits its nature. The fact that the ore is drawn from the stope and worked through the grizzly at the same place permits one man to perform both operations.

Electric firing is used for block-holing bowlders on grizzlies and starting hung-up drawing raises for safety and for eliminating smoke from fuse. The entire active portion of the panel is wired in parallel with a permanent blasting circuit. There are four breaks in this circuit, as follows:

(1) An open clip is installed in a drift set adjacent to each grizzly. The lead wires are attached to this clip.

(2) A similar open clip is placed at the entrance to each grizzly line.

(3) There is an interrupter switch in the supply drift at a safe distance from the last grizzly line.

(4) The firing is done by means of a firing switch at the exit of the stope. As an additional safeguard the wire used to close the open clip at the entrance to each grizzly line is attached to a chain hanging on the opposite side of the grizzly drift. When the wire is in place in the clip the chain from which a blasting danger sign is suspended must be drawn across the grizzly drift.

By the use of electric firing the chute-tapper is able to place his bombs in the most favorable positions in the hung-up fingers, can remove the plank covering over the grizzlies as he retreats, is not required to work in a smoky atmosphere as when using fuse, and has ample time for all operations, so that he is not hurried and does not have to work under strain.

Morenci practice is dictated by the existence of a large proportion of hard, coarsely-breaking ore which has developed three innovations: (1) Drawing through large openings in the sides of the grizzly drifts instead of through timbered chutes; (2) using grizzlies in the stopes with bars spaced 16 inches apart instead of the customary 10 or 12 inches; (3) using one man to draw the ore and also work it through the grizzly.

So much coarse rock occurs in every section that ore can not be drawn through small chute doors satisfactorily. Openings 4 by 6 feet in section are therefore cut in the sides of the grizzly drifts, which connect with the finger raises. The ore flows through these large openings to the grizzlies where it is broken up by sledging or block-holing. At the grizzlies there is enough room to work and to get away if a surge of ore occurs. Wide spacing of grizzly bars reduces sledging and block-holing, although it then becomes necessary to change haulage, hoisting, and coarse-crushing equipment to accommodate large bowlders. The advantage of the wide spacing is reflected in chute-tapping efficiency, which has increased to its present figure of 120 tons per man shift from an average of 37 tons per man shift with 10-inch grizzlies. As a protection against falling throug widely-spaced grizzlies the chute tapper is provided with a safety belt which he always wears while at work on the grizzly. Contrary to the popular impression before this innovation was tried, the belt does not interfere with the output of the workman. When not actively in service all grizzlies are covered with two 10-inch planks. Handholds of substantial chain are installed on both sides of each grizzl

Previous to the introduction of the wide grizzlies it was occasionally impossible to put the ore through as fast as stope conditions dictated. Drawing can now be increased to a rate of 1,200 tons per day from an area of 10,000 square feet whenever it may be found desirable to do so; this means caving at a rate of 1½ feet vertically per day.

In the active portion of a producing panel some ore must be drawn from eac finger every day to prevent local packing. The minimum draw in areas which are being held back for purposes of stope control is 12 tons per finger.

Each process in the preparation for drawing must be designed to conform with the factors governing the efficiency of that operation.

(1) The ore must be completely undercut. If even a small area remains unbroken it acts as a pillar, by which excessive pressure is transmitted from the broken ore to the grizzly level. The layer of ore broken during undercutting must be thick enough to allow caving to begin.

(2) Enough ground must be cut through in the boundary shrinks to permit the blocks to begin caving without delay after undercutting, but not enough to cause the whole block to collapse at once. The continuous stope cut out for the lower 35 feet of the block boundary assists the caving. Leaving part of the ground in place on the boundary above prevents caving too fast, and in large masses.

Morenci ore varies so much that no constant percentage of the area on the boundaries can be used to decide how much shrinkage stoping is to be done. That must be left to the judgment of the operator. It is better to cut out too much, even at the risk of a sudden collapse, than too little and have the block hang up indefinitely. Moreover, sections of unbroken ground on the stope boundaries should not be so great that the line of shearing may diverge from the desired plane.

(3) Drawing points are subjected to great pressure. As far as possible they should be protected from weight in order to permit continuous drawing and to avoid repairs which would interrupt drawing and increase the cost of maintenance.

Weight is resisted most efficiently by leaving as much unbroken ground as possible about the grizzly drifts to act as pillars. This can be accomplished by keeping the size of the grizzly drifts small and by cutting down the number of finger raises that are driven through it. Wide spacing of the fingers is limited by the requirements of good drawing, which prescribe spacing them as closely as possible. The result is a compromise spacing such as will yield the best returns.

Pressure on the grizzly drifts is affected by the rate of drawing. If it is too slow, weight on the grizzly drifts will gradually accumulate, while rapid drawing relieves that pressure.

(4) The grizzly bar spacings limiting the size of bowlders that pass through them should be as wide as the subsequent operations of hoisting, haulage, etc., will permit. When the ore to be handled is naturally coarse the spacings should be planned to handle large bowlders conveniently. Chute-tapping labor is thus reduced and the speed of drawing increased.

(5) Haulage raises should have sufficient storage capacity to make the operations of ore-drawing and haulage independent of each other.

The tendency in stoping operations at Morenci, as at other mines in the Southwest, is toward higher lifts, which reduce preparatory or development costs. In 1928 the average stoping lift was 164 feet, while on the next lower stoping horizon this will increase to 270 feet.

An increase in the height of the ore column above 250 feet reduces the cost of stoping very little. It increases the total length of time required for the ore to reach the mill after being broken in the stope. This delay in drawing may be detrimental to certain ores that oxidize rapidly for concentration by gravity or flotation. There may be a slight filming of the sulphide surfaces that interferes with flotation without an appreciable increase in the amount of acid-soluble copper

Method of Controlling the Draw

When a new stope is ready for production, the plans and sections for it are assembled by the stope-control department. The plan is a special map of the stope showing each finger and corresponding haulage raise. On the plan the line of fingers is numbered in one direction to show the page number on the section; in the other direction the numbers designate the grizzly lines, as shown in Figure 16. Thus "4-96" is used to designate the first finger on the south side of the bottom row. This method identifies the finger on the office records and also gives its location underground; several fingers branch from one raise. After the plan is completed the tonnage tributary to each finger is calculated from the ore outlines shown on the assay maps. These tonnages are recorded on the finger map and are adjusted to agree with the ore reserves. From these tonnages, longitudinal sections are made. Each page of sections represents a line of fingers (fig. 16). These sections are based on broken ore volumes as distinguished from rock in place. The increase in volume is termed the swell. The ore in place runs 12 cubic feet to the ton while the broken rock is considered to be 20 cubic feet to the ton, so that each finger in the section is increased 40 per cent to allow for the breaking up of the rock. These plans and sections are made on a scale of 1 inch to 20 feet. In the present stopes, where the fingers are spaced on 14-foot centers in one direction and 18 2/3-foot centers in the other direction, the tonnage scale runs 13.07 tons to the foot. The tonnage is posted on the sections in different colors, according to the month the ore was drawn. The stope is controlled primarily from these sections. They show (1) the angle of retreat; (2) when a finger is being pulled too fast; (3) when a finger is not being pulled at all; and (4) when the stope as a whole is being pulled too slowly or too rapidly.

If a pipe of cap rock reaches a draw point in advance of the upper boundary of the ore body, the finger is sealed and ore is drawn from the surrounding raises until the waste pipe is broken. Close spacing of draw points facilitates the breaking up of pipes of waste; if the draw points are too far apart they do not assist each other.

The proper rate of draw is dependent on the rate at which the unbroken ore caves. If the ore is drawn too rapidly too large an open space will appear between the broken and unbroken ore. In such case the ore is likely to break in large masses. If it is drawn too slowly the swell of the broken ore will partly support the arch above and delay caving.

After caving reaches the top of the ore there appears to be no limit of drawing speed other than the ability to put the ore through the grizzlies and preserve a proper relative movement of the ore and waste capping to the chutes. If a heavy spot appears in the stope, more rapid drawing at that area relieves pressure on the grizzly level.

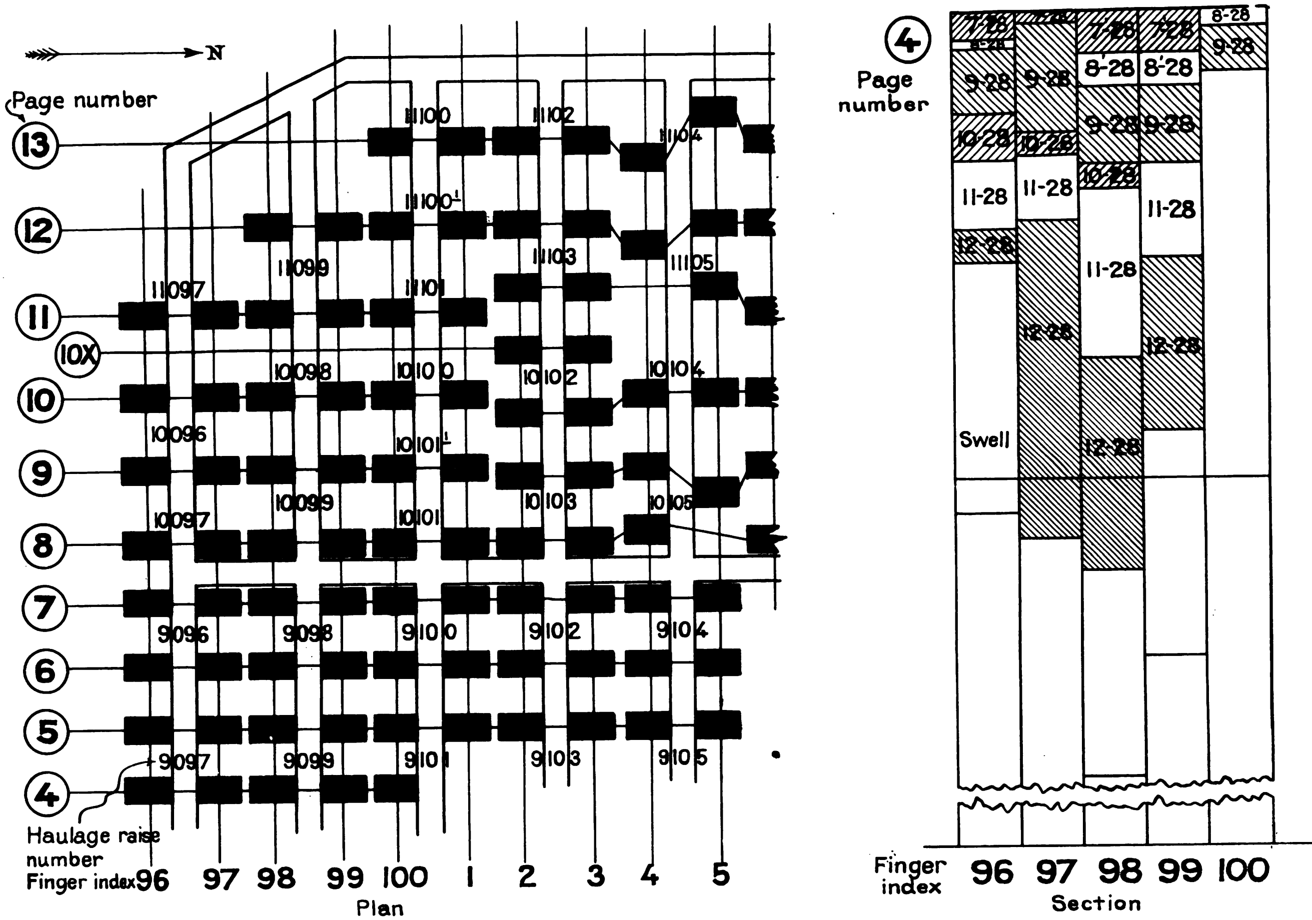


FIGURE 16.—Stope control

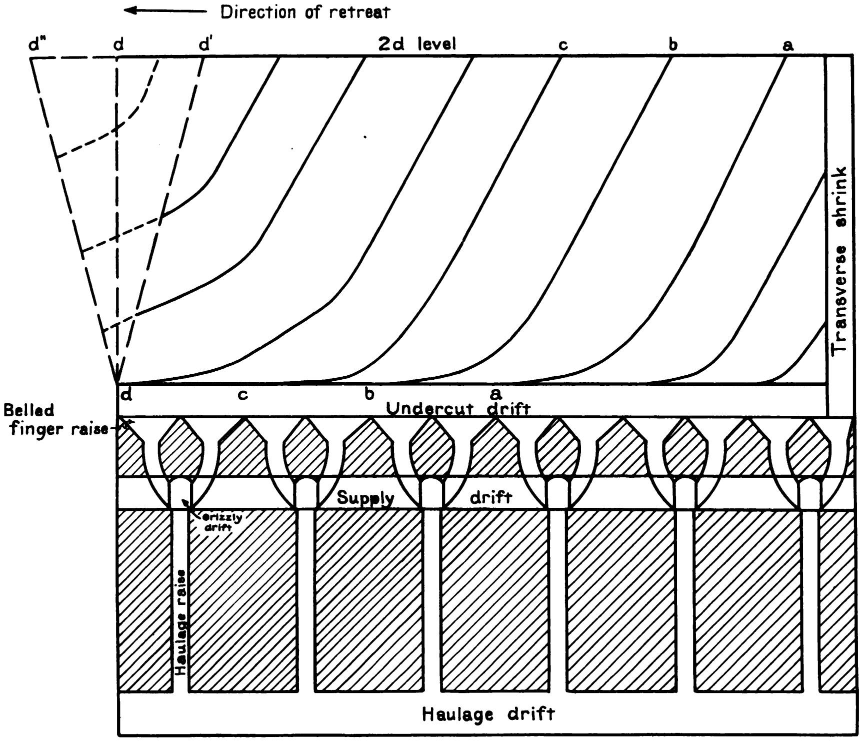


FIGURE 17.—Typical section showing progressive stages of caving. Looking at right angle to line of retreat

When the ore in a section is nearly all drawn off, the rate of drawing is decreased to give time for sampling doubtful chutes and obtaining the maximum recovery of clean ore.

During the initial stages of drawing the angle of retreat is 60°. As the stope is extended the angle is decreased to approximately 50° or to even a smaller angle if weight does not interfere. This angle is maintained to the completion of the stope.

As the undercutting advances, a certain tonnage of the most recently undercut ore is held back undrawn in the reserve column. The swell in this section is released for drawing at a slow rate. By this means the entire block is broken up more finely in advance of the regular draw. The reserve of broken ore continually varies according to the relative amount of ore undercut and drawn.

In each stope and on each shift there is a chute checker whose duty is to see that the ore is drawn according to the draw sheet which is issued to him each day by the control department. He makes a tonnage report for each finger worked during his shift and also takes samples from each finger. He turns in a report from the transportation department showing the number of cars pulled from his stope during the shift. He closes off the fingers marked "waste" and also makes a note of any new fingers that start running waste during his shift and reports this to the office. The chute checker keeps track of the haulage chutes that are full and delivers a list of these to the transportation crew at regular intervals. This list is dropped from the grizzly level to the haulage level by way of a 3-inch pipe that runs through an old chute in an adjacent section of each stope.

When the reports are received in the office the cars are apportioned accord ing to the fingers worked. The tonnages are then posted on the finger sections anc new draw sheets made out. The draw sheet shows the total number of tons released in each finger, the fingers to be closed off, and the maximum daily rate of draw for each finger. The results from the finger samples are posted on a sheet showing the number of cars drawn from that finger during the day. These assays are all averaged together according to their tonnages to give a composite result for the entire stope production for the day.

Ore drawing is an art. The engineer in charge of the stope-control department starts with a knowledge of the quantity, position, and assay grade of the ore in place. From the quantity of ore drawn, and its grade, the weight on the drawing section, and the behavior of certain finger raises, he is able to form an idea of what is happening in the mass of broken ore above, how it is moving, and how to plan for its extraction.

The stope-control department must rely on the full support of the mine foreman and his staff. It must be understood, however, that the daily production must be made and the control department in turn must assist the miners. Unless there is entire cooperation something will suffer.

One of the most interesting phases of caving operations is the rock-mechanics of the caving action. There are many theories as to the details of this action which are usually prompted by observations of individual stopes in a particular mining district. The Morenci conception of a normal case is illustrated in the longitudinal section (fig. 17), which is typical of uniform ground. Any major planes or areas of weakness would of course induce different results from those illustrated.

The actual undercutting is delineated in Figure 17. Caving is initiated along a rather flat inclined brow which often approximates a slope of 30°. This steepens against the caved ore to perhaps 60°. With sufficient time interval this brow stabilizes in an almost vertical position at the extreme edge of the undercut. It may (1) project slightly over the caved ore, (2) stand exactly perpendicular, or (3) even extend at a small angle beyond the basal undercut (d', d, d'' in fig. 17).

The rate at which the fragments of ore, locally termed raveling, drop down from the unbroken mass depends upon the tensile, shearing, and crushing strength of the ore. The strength in turn is affected by such physical characteristics as fracturing, faulting, jointing, texture, hardness, chemical composition, and alteration. The actual rate in a vertical distance may vary from 0 to 20 or more feet per day.

This brow probably constitutes a cantilever whose fulcrum is immediately beyond the edge of the undercut along the line of retreat. In certain hard ground at Morenci, to protect the advance stope development from closing in as a result of concentrated weight at this fulcrum, shallow transverse shrinkage stopes were deiven entirely across the stoping panel. In softer ground such shrinks are unnecessary.

When the transverse shrinks are driven, the stope resembles the rectangula blocks, in use elsewhere, that are cut off on all sides and in which the ore is drawn down horizontally. At Morenci the proper angle of retreat is maintained.

To prevent undue weight a slower initial rate of draw at the edge of the undercut towards the line of retreat has proved beneficial. By this treatment the ore is also broken in smaller pieces.

The time element is an important factor in breaking ground fine. Excessiv speed in undercutting and drawing, or transverse and boundary shrinks driven too high, will produce coarse-running ore.

A series of boundary shrink drifts with intervening horizontal level pilla gives the proper delay in caving in soft ground, but there are few blocks in Morenci where such drifts can be used. The ground is usually so hard that it bridges over these drifts and all caving action is stopped, unless enough ground is cut out to make the caving positive. In Morenci ground the horizontal pillars must be cut down to such narrow dimensions that this method is not economical.

However, the delayed effect has been obtained by the saw-tooth shrink, as described. The first ore drawn after the actual undercut ore is removed, is usually the coarsest. A grinding action is later established in the higher ore column, which produces finer ore. The weight of the overburden undoubtedly assists this action. In one case at Morenci, the ore in a stope, with a lift of about 150 feet, extended almost to the surface without the usual leached capping. This block produced very coarse ore.

In a wide stope it has been observed at Morenci that the greatest weight on the grizzly level pillars is in the center of the panel. The resulting weight is proportioned similarly to the deflection of a uniformly loaded beam supported at the ends. By narrowing the section from the original 165 feet to the present 112 feet this effect was diminished. It is thought that the side friction of the moving ore along the walls of the stope acts as a retardant.

Dilution.- Owing to uncertainties in tonnage and assay grade of certain blocks in older sections of the mine because of incomplete information, the data on dilution at Morenci are not considered entirely reliable and hence are not included at this time. The ore, breaking very coarse, draws down quite uniformly and has but little tendency to pipe. The result has been exceptionally clean ore with very little dilution.

The method of computing dilution is the same as is used at the Ruth mine of the Nevada Consolidated Copper Co. In this method the capping is assumed to be barren. The percentage of copper extracted as compared to the expectancy equals the percentage of clean ore shipped. The tons of capping shipped can then be calculated. The percentage of capping shipped is the dilution.

UNDERGROUND TRANSPORTATION

About 5,000 tons of ore per day was mined during February, 1929. The ore from the stopes is loaded into 6 or 8 ton cars and pulled by electric locomotives about three-fourths of a mile to No. 6 shaft, where it is hoisted to the mill bins.

The haulage department is operated under separate head; haulage is continuous for 24 hours per day six days per week. The mill operates seven days per week.

Since each "line" or row of grizzlies on the grizzly level feeds the chutes located in two or more haulage drifts, increasing the number of lines producing ore at one time has increased the number of chutes that may be drawn in each haulage drift, thus obviating some switching in the collection of a trainload of ore.

The loop system of haulage is used (fig. 14).

The trains consist of 15 eight-ton cars or 20 six-ton cars. Two haulage crews are used on the main line and two crews in the stoping area. A train can be loaded in about the same time that a round trip can be made to the shaft. Each main-line train makes 9 to 13 trips per shift. A train of 15 cars is usually unloaded in 7 minutes. The full-load speed of the train is 6 miles per hour.

The head brakeman of each train fills out a printed form showing the number of cars pulled from each chute as it is drawn. This report is given to the stope-control department in the engineering office at the close of the day. This record is checked against the chutes reported full by the chute checker. The actual tonnage is allocated to its proper chute by applying a car factor obtained each day by weighing all loaded and empty trains on platform scales which are situated near the shaft.

Loading at the chutes, switching from chute to chute in collecting a trainload of ore, and dumping at the shaft occuy about three-fourths of the operating train time.

By widening the chute doors from 30 inches to 42 inches and increasing their area, the speed of loading was nearly doubled. Large chute doors are particularly important when there is coarse ore.

HOISTING

Only one class of ore is mined on company account. All of it--about 140,000 tons per month--is hoisted through No. 6 shaft from the 5th level.

The ore is hoisted in two 4.6-ton skips run in balance. The distance from the skip loading station to the dumping position is 512 feet. The average time for a round trip of one skip is 2.0 minutes. The average number of skips hoisted per day including Sundays is 1,250.

Waste from development work is hoisted through the Liverpool shaft. Men and supplies for the lower levels are also handled at this shaft.

The Joy shaft is used for handling men and supplies to the stoping area of the mine.

VENTILATION

The mine is mechanically ventilated by two centrifugal fans that have forward-curved blades, double inlets, and horizontal top discharge. They are operate at 340 r.p.m. by belt drive from 100- hp., 2,000-volt induction motors. The rotors are 5.5 feet in diameter and 5.4 feet wide. The fans are situated at opposite ends of the ore deposit at the portals of adits connected by upcast raise with the producing areas. They operate as non-reversing exhausters. One fan exhausts 89,000 cubic feet per minute and the other 79,000. The fans have separate air returns of high frictional resistance and common intakes of low resistance.

The two lobes of the ore body are separately ventilated because of anticipated difficulty in maintaining connection with a common air return. A fan exactly similar to the two surface fans, installed on the No. 5 level in a by-pass near the Liverpool shaft, is maintained in operating condition for fire emergencie

Long development headings are ventilated by the use of small high-speed blower fans of various types, and canvass tubing. Two fans in a station several hundred feet from the fresh-air intake are used in combination for this purpose. The smaller fan is situated on the fresh-air side of the station and blows a large part of the intake air coming in through the drift to the face of the heading through 12-inch canvas tubing. The larger fan exhausts from the face of the heading and discharges the exhaust air through a 16-inch canvas tubing into the nearest return-air opening. By this arrangement some intake air is short-circuited into the exhaust air but no exhaust air can be returned to the working face.

The mine is cool throughout, both winter and summer. During the winter season it is necessary to avoid producing high-velocity air currents in working zones in order to avoid uncomfortably cold conditions. The rock temperature in the lowest development headings--a drainage tunnel on the No. 18 level 900 feet below the adit level--is, however, 76°, so that the coolness of the workings is due to the shallow depth and lack of any rapid degree of oxidation in the caved and abandoned areas adjacent to working areas.

Air samples show that the quality of the air, as to chemical composition, is good throughout the mine.

The ventilation system is planned to draw dust and foul air out of the mine as directly as possible without carrying it through working places.

DRAINAGE

The only pumping in the present workings of the Morenci Branch is in the bottom or 14th level of the Liverpool shaft. This level has been driven from the Liverpool shaft past the Joy shaft to No. 6 shaft--a total distance of 4,000 feet. As the grade is slightly downhill in the direction of driving from a point 1,000 feet from the Liverpool, water tends to collect and must be forced back to the Liverpool pumps.

The pumps used for this purpose are two $1\frac{1}{2}$-inch centrifugals, driven by 5-hp. direct-current motors. One of these is pumping a distance of 3,000 feet against a 10-foot head. The other handles less from a drift heading which is not advancing at present.

Two electrically-driven triplex pumps are in use at the Liverpool shaft. One takes water from a sump at the 14th level, 725 feet below the collar of the shaft. The other operates from a sump at the bottom of the shaft, 830 feet below the collar. A flow of 50 gallons per minute is being pumped at present from these two points to the surface.

WAGE SYSTEM

For the year 1928, 72 per cent of all labor in the mining department was paid on bonus or contract systems. The other 28 per cent was on day's pay.

Contracts

A standard set of contract prices is in effect throughout the mine. The foremen who let the contracts have copies of the contract schedules. These contract prices cover such work as drifting, raising, and winzing--both driving and timbering--laying track, digging ditches, undercutting and shrinking in stopes, and salvaging mining material. The rates are followed very closely and seldom changed. Unusual conditions are stated on the contract form and authority for changes from the standard must be obtained from the mine superintendent before a contract is let. All measurements are made on the 15th and 30th of each month, and the earnings are computed by the engineering department. All men on contract are guaranteed day's pay at the prevailing wage. The time and class of work done by each man on a contract is recorded by the shift boss.

The manner in which this contract system differs from systems employed at many mines lies in the way in which the proceeds of a contract are shared among the workmen participating. The earnings may then be divided by any one of four methods: (1) An equal division of earnings to all men concerned in the contract; (2) the payment of day's pay rates to all according to their respective day's pay rates, with an equal division of the surplus; (3) payment of day's pay rates to all, then a division of the surplus in proportion to his day's pay rate; (4) straight payment to each man for the work that he does, having separate prices on timbering, mining, mucking, etc. The choice of the method of dividing the proceeds of the contract is left to the discretion of the foreman, who is familiar with the work, its conditions, and the desires of the men.

Bonus

A bonus system is applied to men working in stopes and on transportation crews. In arriving at the bonus rates past daily performances of men employed at various classes of work have been averaged to determine a "base" for each class. This base represents the amount of money that would be paid if a man does an average day's work, and corresponds to a day's pay. This figure divided by the number of tons representing an average day's work is called the "base cost per ton" (or other unit). If a man exceeds an average day's work he is paid one-half his "base cost per ton" for each unit by which he exceeds average day's work.

The base in each working place is subject to change at the beginning of any period if conditions should warrant it.

At the drill-sharpening shop the proposition was made to the men that one-half of any saving in labor effected by a reduction in the force would be divided equally among the men in the shop. This ruling had the effect of raising the quality of the bits turned out because with a higher standard of bit there was less work in the shop and less men were required. The base number of men is subject to change at the discretion of the mine superintendent when any of the operating conditions undergo changes, such as variation in amount of work done in the mine, change in shop practice, change in shop equipment, or change in scale of operations. The drill-shop foreman was not included in this bonus.

The mine shift bosses do not receive a production bonus but do participate in an accident-prevention bonus based on a graduated scale for each 1,000 shifts supervised without an accident.

ACCIDENT PREVENTION

Safety of workmen is considered a major operating problem. The management insists that accident prevention be considered before production or costs. Plans for all work are first considered and approved from the standpoint of safety.

The safety organization consists of a branch safety engineer and a mine safety inspector. Safety inspection committees, consisting of workmen and the safety inspector, make periodical inspections and recommendations for eliminating hazards whether in practices or equipment. The work involved in carrying out these recommendations, when approved by the superintendent, is given precedence over all other operations.

General safety meetings attended by all the workmen and bosses in each section of the mine, are held at regular intervals on company time for the discussion of safer practices.

First aid is taught to all new employees. All employees are encouraged by yearly competitions and prizes to review their first-aid traing.

Two mine rescue teams are kept in regular training and a third team is kept in reserve training.

The mining staff at regular intervals is given special instruction in handling underground fires to prevent loss of life and destruction of property.

COST OF OPERATION

Table 1.- Summary of costs in units of labor and supplies, Humboldt mine, Morenci Branch, Phelps Dodge Corporation.

(Period covered: 1927 - 1928. Tons of ore hoisted: 140,000 per month. Mining method: Undercut block-caving.)

MINING

Year	Tons mined	Tons per man			Pounds powder per ton	Board feet timber per ton
		Stoping[1]	Mine pay roll[2]	Overall[3]		
1927	1,136,339	40.35	10.00	9.07	0.24	0.23
1928	1,483,984	62.76	11.84	10.45	.19	.25

[1] Includes: (1) Boundary shrinkage stopes; (2) undercutting; (3) belling fingers; (4) chute tapping; and (5) stope repairs, but is exclusive of all development work.

[2] Tons per man for shifts on mine pay roll, including men on salaries.

[3] Tons per man for shifts charged to mining, including all labor from other departments, such as mechanical, electrical, carpenter, and other surface departments occupied on work chargeable to mining.

The output per man on present production and work associated with it, including all shifts on the mine pay roll with salaried employees for the year 1928, is approximately 18.5 tons. This excludes all lower level development which is for the purpose of bringing into production the next lower lift.

Table 2.- Summary of costs in units of labor and supplies, Humbolt mine, Morenci Branch, Phelps Dodge Corporation.

(Period covered: 1927 - 1928. Tons of ore hoisted: 140,000 per month. Mining method: Undercut block-caving.)

DEVELOPMENT

	Size of heading, feet	Feet per man shift		Pounds powder per foot		Board feet timber per foot	
		1928	1927	1928	1927	1928	1927
Grizzly drifts (raw)	4 by 6)	2.81	2.58	5.12	5.85	- -	- -
Supply drifts (raw)	6 by 8)			6.43	6.13	- -	- -
Motor drifts (raw)	8 by 9)	.98	1.26	10.87	8.01	- -	- -
Motor drifts (timbered)	10 by10)			9.37	7.50	- -	- -
Averages, total driving		1.52	1.93	- -	- -	- -	- -
Timbering drifts	6 by 8 or less	- -	- -	.82	.32	58.44	52.94
Timbering drifts	8 by 9 or over	- -	- -	1.08	.63	107.88	100.98
Averages, total driving and timbering		1.00	1.36	- -	- -	- -	- -
Finger raises (raw)	4½ by 4½)	5.71	4.84	3.82	4.83	- -	- -
Short haulage raises (raw)	4 by 6)			3.97	3.94	- -	- -
Long raises with dividers	4 by 7	1.47	1.49	5.97	8.25	28.18	16.54
Cribbed raises	4 by 7	1.10	1.73	4.83	2.96	84.20	62.71
Morenci slides	- -	- -	1.81	- -	5.15	- -	74.79
Averages, raises		3.91	[illegible].48	- -	- -	- -	
Winzes		.42	- -	3.81	- -	52.53	- -
Grand Averages, all driving and timbering		1.71	2.35	6.36	6.91	61.53	40.28

Table 3.- Estimated cost of preparatory work for a block containing 120,000 tons, Morenci Branch, Phelps Dodge Corporation [a]

	Tons	Shifts	Units	Cost per unit				Total cost, labor, timber and explosives	
				Labor	Timber	Explosive	Total	Amount	Cost per ton
Supply drifts	600	87	130 ft.	$ 4.30	$ 2.20	$1.15	$ 7.65	$ 994.50	
Grizzly drifts	1,500	228	390 ft.	3.32	1.63	1.10	6.05	2,359.50	
Grizzlies	200	302	14 [b]	118.60	180.63	12.12	311.35	4,358.90	
Fingers	1,500	170	830 ft.	1.03	- -	.89	1.92	1,593.60	
Pilot raises	200	54	80 ft.	3.14	.73	1.36	5.23	418.40	
Shrink drifts	500	78	160 ft.	2.59	- -	1.07	3.66	585.60	
Totals	4,500	919	1,590 ft.					10,310.50	$ .086
Shrinkage stopes (tonnage used is swell only)	1,500	140	9,000s.ft.	.10	- -	.08	.18	1,620.00	
Undercutting (tonnage used is from dog holes only)	3,300	336	10,000s.ft.	.28	- -	.22	.50	5,000.00	
Totals	4,800	476						6,620.00	.055
Total above grizzly level	9,300	1,395	1,590 ft.	- -	- -	- -	- -	16,930.50	.141
Haulage drifts	2,500	286	330 ft.	5.86	1.99	1.42	9.27	3,059.10	
Loading chutes	300	119	7	73.04	104.70	5.02	182.76	1,279.32	
Haulage raises	1,300	207	600 ft.	1.56	.30	.72	2.58	1,548.00	
Total below grizzly lev	4,100	612	930 ft.	- -	- -	- -	- -	5,886.42	.049
Total development exc. of shrinks & undercut	8,600	1,531	2,520 ft.	- -	- -	- -	- -	16,196.92	.135
Grand total	13,400	2,007	2,520 ft.	- -	- -	- -	- -	22,816.92	.190

[a] Area = 10,000 square feet. Lift = 145 feet. Grizzly Drifts at 28-foot centers and Fingers at 14-foot by 18-foot, 8-inch centers. This estimate was originally made (April, 1928) to assist in planning operations for another mine of the Phelps Dodge Corporation.

[b] 7 single and 7 double grizzlies.

NOTE: All costs per ton are calculated on a basis of 120,000 tons production, unless otherwise noted.

Table 3.- Estimated cost of preparatory work for a block containing 120,000 tons, Morenci Branch, Phelps Dodge Corporation (Continued)

	Tramming		Hoisting		Sanitation		Ventilation		Assaying and sampling	
	Amount	Cost per ton	Amount	Cost per ton	Amount	Cost per ton	Amount	Cost per ton	Amount	Cost per ton
Supply drifts	$ 71.40		$35.40		$ 4.35		$13.92		$20.87	
Grizzly drifts	178.50		88.50		11.40		36.48		62.59	
Grizzlies	23.80		11.80		15.10		48.32		- -	
Fingers	178.50		88.50		8.50		27.20		- -	
Pilot raises	23.80		11.80		2.70		8.64		25.68	
Shrink drifts	59.50		29.50		3.90		12.48		25.68	
Total	535.50	$ .004	265.50	$ .002	45.95	$ .000	147.04	$ .001	134.82	$ .001
Shrinkage stopes (tonnage used is from swell only) ...	178.50		88.50		7.00		22.40		288.90	
Undercutting (tonnage used is from dog holes only)	392.70		194.70		16.80		53.76		- -	
Total	571.20	$ .005	283.20	$ .002	23.80	$ .000	76.16	$ .001	288.90	$ .002
Total above grizzly level	1,106.70	$ .009	548.70	$ .005	69.75	$ .001	223.20	$ .002	423.72	$ .004
Haulage drifts	297.50		147.50		14.30		45.76		52.96	
Loading chutes	35.70		17.70		5.95		19.04		- -	
Haulage raises	154.70		76.70		10.35		33.12		192.60	
Total below grizzly level	487.90	$ .004	241.90	$ .002	30.60	$ .000	97.92	$ .001	245.56	$ .002
Total development exclusive of shrinks and undercutting ...	1,023.40	$ .008	507.40	$ .004	76.55	$ .001	244.96	$ .002	380.38	$ .003
Grand total	1,594.60	$ .013	790.60	$ .007	100.35	$ .001	321.12	$ .003	669.28	$ .006

Table 3.- Estimated cost of preparatory work for a block containing 120,000 tons, Morenci Branch, Phelps Dodge Corporation (Concluded)

	Drills and tools		Operating supplies		Mine department expense		Grand total	
	Amount	Cost per ton	Amount	Cost per ton	Amount	Cost per ton	Amount	Cost per ton
Supply drifts	$ 179.40		$14.30		$ 78.00		$	
Grizzly drifts	538.20		42.90		195.00			
Grizzlies	463.12		60.40		26.00			
Fingers	1,145.40		91.30		195.00			
Pilot raises	110.40		8.80		26.00			
Shrink drifts	220.80		17.60		65.00			
Total	2,657.32	$ .022	235.30	$ .002	585.00	$ .005	14,916.93	$ .124
Shrinkage stope (tonnage used is swell only)	1,100.40		143.00		195.00			
Undercutting (tonnage used is from dog holes only)	2,640.96		121.00		429.00			
Total	3,741.36	$ .031	264.00	$ .002	624.00	$ .005	12,492.62	$ .104
Total above grizzly level	6,398.68	.053	499.30	.004	1,209.00	.010	27,409.55	.228
Haulage drifts	455.40		36.30		325.00			
Loading chutes	77.98		23.80		39.00			
Haulage raises	828.00		66.00		169.00			
Total below grizzly level	1,361.38	$ .011	126.10	$ .001	533.00	$ .004	9,010.78	$ .075
Total development exclusive of shrinks and undercutting	4,018.70	$ .033	361.40	$ .003	1,118.00	$ .009	23,927.71	$ .199
Grand total	7,760.06	$ .064	625.40	$ .005	1,742.00	$ .014	36,420.33	$ .303